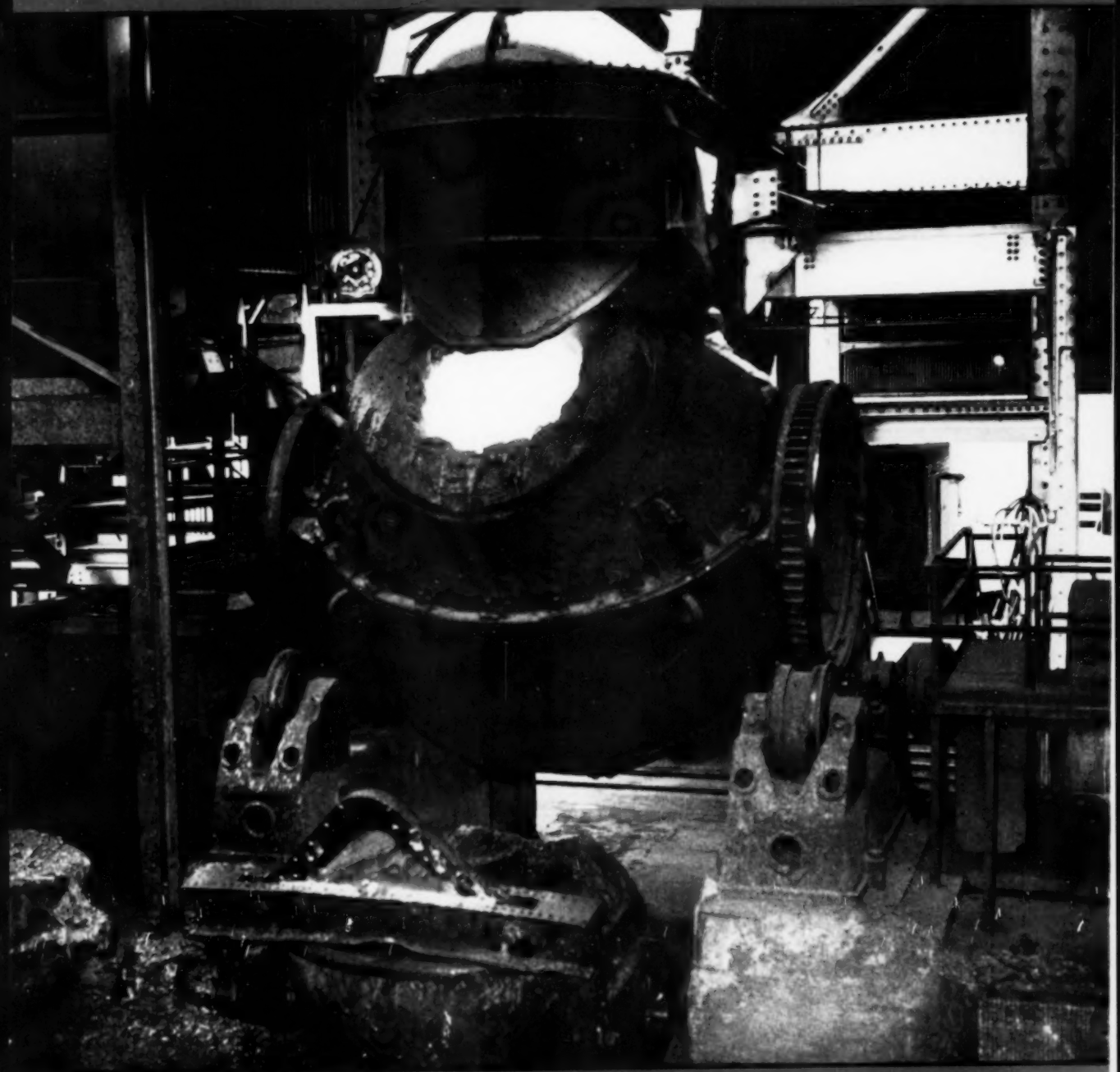
MAY, 1953
IN TWO SECTIONS — SECTION ONE

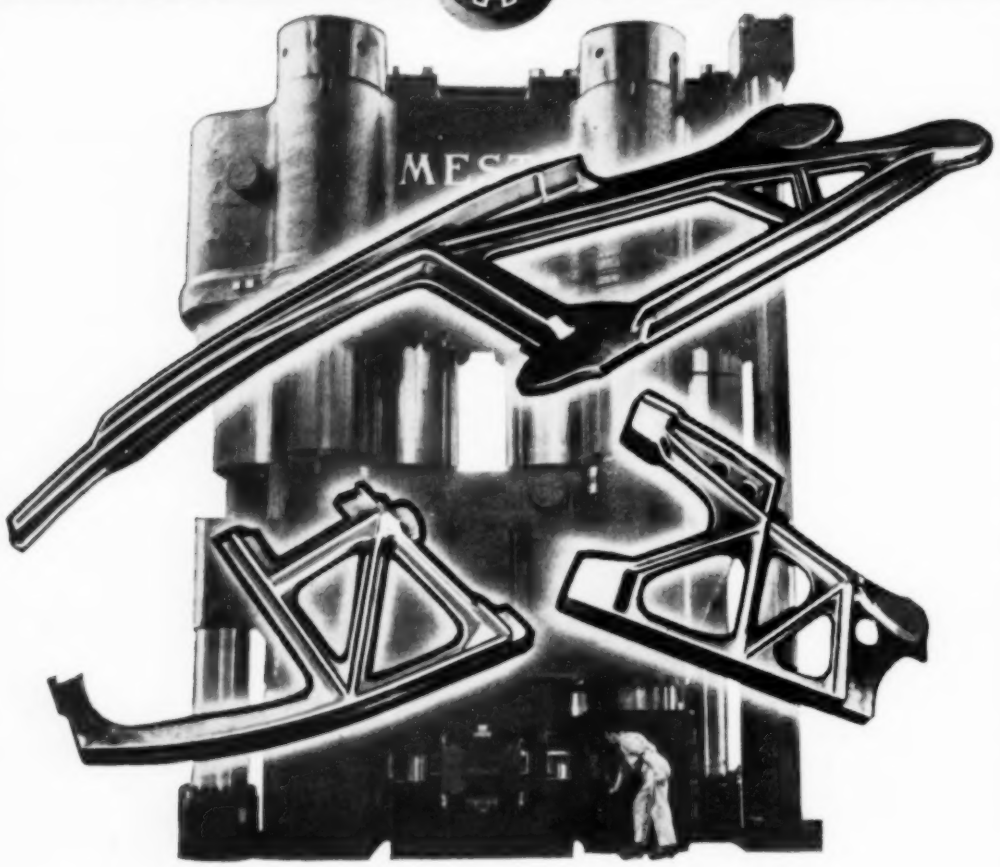
Journal of METALS



May JOURNAL OF METALS Highlights:

"Asarco Rehabilitates Copper Refinery" - p. 616

"Operations at New Cornelia Copper Smelter of Phelps Dodge Corporation" - p. 633



Greater Size and Speed in Aircraft have created engineering problems, the solution of which has required larger and larger forgings of high-strength aluminum alloy. *Examples* shown above are forged structural members used in a modern military bomber, the largest more than seven feet over all. These are forged on an 18,000-ton press, the biggest ever built in this country.

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Journal of METALS

VOL. 5, NO. 5 MAY, 1953

IN TWO SECTIONS — SECTION ONE

COVER

In a departure from conventional converter practice at Phelps Dodge Ajo smelter, blister copper is overblown in a separate oxidizing and holding vessel. An old 12-ft Great Falls converter was slightly remodeled and installed for this purpose. See article on p. 633.

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J. W. Byrkit (p. 633) is a Member of the Arizona Section, AIME. Mr. Byrkit was employed by the United Verde Copper Co., Clarkdale, Ariz. as smelter laborer in 1924. He held various operating and supervisory positions in the smelter and mill and was research engineer for five years. The firm was acquired by Phelps Dodge in 1935. From 1937 to 1942 he was concentrator superintendent, and then became superintendent, smelter div., United Verde Branch. In 1950 Mr. Byrkit was named smelter superintendent at the New Cornelia Branch, a position he presently holds. A native of Kokomo, Ind., he attended DePauw University, University of Nevada, and Stanford University. Mr. Byrkit enjoys photography as his hobby and also plays golf.

J. W. BYRKIT

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G. B. Paulding (p. 616), superintendent of power, American Smelting & Refining Co., Perth Amboy, was born in Brooklyn. A graduate of the Colorado School of Mines in 1939, he was employed by New Jersey Zinc Co. from 1939 to 1943. During World War II, Mr. Paulding was an Aviation Ordnance Officer in the Navy. Upon his discharge he joined AS&R. Now residing at Garwood, N. J., he likes fishing in his spare time.

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J. C. FISHER

H. K. SPAULDING

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You Chao Liu (p. 667) born in Shanghai, received his education in China and the United States. He attended the National Chiao-Tung University in 1947, Yale University in 1950 and 1952. Dr. Liu is presently a research associate in the engineering research div., New York University. A member of the ASM, he is a Junior Member of the New York Local Section, AIME.

Laurence Delisle Pellier (p. 660) was born in Paris but attended Stevens Institute of Technology and City College of New York. In 1940 Mrs. Pellier was a research associate in the powder metallurgy dept. at Stevens Institute. She then joined Sylvania Electric Products Inc., as senior engineer in the metallurgy dept. Now associated with American Cyanamid Co., Stamford she is a metallurgist working on corrosion problems and various other subjects. Mrs. Pellier is a member of the New York Local Section of AIME, Metal Science Club, Societe Francaise de Metallurgie, and several other professional societies.

L. D. PELLIER

W. CRAFTS

Walter Crafts (p. 649) is associate director of research, Union Carbide & Carbon Research Laboratories, a position he has held since 1952. Mr. Crafts is a native of Oberlin, Ohio. He received his A.B. from Yale (1924) and M.S. from Massachusetts Institute of Technology (1926). He was employed by Illinois Steel Co. from 1926 to 1929. Mr. Crafts joined the Union Carbide & Carbon Research Laboratories as a research metallurgist. In 1946 he was named chief metallurgist for the laboratories. Woodworking, music, and fishing are the hobbies of this resident of Niagara Falls, N. Y.

R. J. Van Thyne (p. 670) has been with Armour Research Foundation since graduation from the Illinois Institute of Technology (B.S.), where he is presently an associate metallurgist. Fishing and gardening are his hobbies.

R. J. VAN THYNE

J. W. EVANS

J. W. Evans (p. 655) is presently physical chemist in the research dept. of the International Nickel Co., Copper Cliff, Ontario. Dr. Evans is a graduate of Queen Mary College, University of London, and the Imperial College, both of London. In 1949 he was employed in the research bureau of the British Iron & Steel Assn., London. He joined the staff of the Ontario Research Council, University of Toronto in 1951 and a year later became associated with the International Nickel concern. Music, old churches, and walking are the hobbies of this author.

H. H. Kellogg (p. 643) has previously authored several papers for AIME. Prof. Kellogg is a native New Yorker and attended Columbia University; B.S. (1941), M. S. (1942). In 1942 he was associated with Pennsylvania State College as chief of the div. of mineral preparation. He then joined the faculty at Columbia and is associate professor of extractive metallurgy. In his free time, Prof. Kellogg enjoys reading and music. He is a member of the Extractive Metallurgy Div., AIME and holds membership in the ACS and Electrochemical Society.

G. H. Weis (p. 616) is assistant manager, American Smelting & Refining Co., Perth Amboy, N. J. From 1928 to 1931 he was foreman of copper refining at AS&R, Baltimore. He was transferred to Perth Amboy in 1931 as dept. head, copper converting. During World War II Mr. Weis was a navigator on aircraft carrier with the Navy. His present rank is Captain USNR. For two years he was ordnance production officer,

Naval Ammunition Depot. He returned to AS&R in 1946 as assistant general superintendent, Perth Amboy. Mr. Weis was appointed general plant superintendent in 1948 and in 1950 became assistant plant manager. Mr. Weis received his B.S. from the U.S. Naval Academy. Among his hobbies he counts golf and music.

D. C. Hilty (p. 649) is head of the dept. of steel research at the Union Carbide & Carbon Research Laboratories, Niagara Falls, N. Y. Born in Apollo, Pa., he is a graduate of Pennsylvania State College (B.Sc., 1935). Mr. Hilty was silicon steel metallurgist for Carnegie-Illinois Steel Corp. from 1935 to 1940. During World War II he was a lieutenant in the Army Engineers, stationed in Iceland. After his discharge in 1943, Mr. Hilty joined UCC as research metallurgist. Boating on Lake Ontario and fishing are enjoyed by this AIME Member.

W. R. Hibbard, Jr. (p. 672) has served as Lieutenant Commander, Bureau of Ships, assistant and associate professor of metallurgy at Yale University and is now research associate for General Electric Co. Tennis, golf, and gardening occupy the spare time of this Member of AIME. Dr. Hibbard was born in Bridgeport, Conn. and is a graduate of Wesleyan University and Yale University.

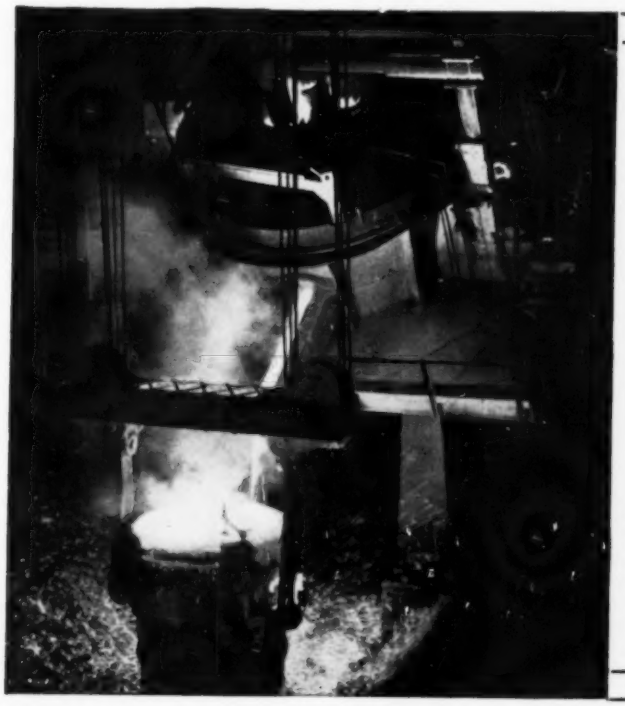
C. F. Baes, Jr. (p. 643) is a research chemist with the Oak Ridge National Laboratory, Oak Ridge, Tenn. Born in Cleveland, Dr. Baes received his B.Sc. (1946) from Rutgers University and his Ph. D. (1950) from the University of Southern California. From 1950 to 1951 he did post doctoral research at the Columbia University School of Mines. As his hobbies, he enjoys bowling and archery.

C. Dean Starr (p. 654) received his B.S. from the South Dakota School of Mines and M.S. and Ph.D. from the University of Utah. He was associated with the A. C. Spark Plug Co., Flint, Mich. as a junior metallurgist and then as spectographer. He has been a research engineer at the University of California since 1950. Woodwork, ceramics, leathercraft, and golf occupy Mr. Starr in his spare time. A native of Tulare, S. D. he now resides at Walnut Creek, Calif.

Louis Silverman (p. 631) has been senior metallurgist at Radio Corp. of America, Harrison, N. J. since 1945. Born in Poland and reared in Detroit, Mr. Silverman is a graduate of Wayne University (B.S., 1941) and Stevens Institute of Technology (M.S., 1950). He had been employed by the Bowan Products Corp., Ecorse, Mich. as a quality superintendent from 1941 to 1945. Hiking and model shipbuilding are Mr. Silverman's hobbies.

H. E. Stauss (p. 653) is head of the electric and magnetic alloys branch, Naval Research Laboratory, Washington, D. C. An alumnus of Washington University and University of California, he holds A.B., M.A., and Ph.D. degrees. Dr. Stauss had been an instructor of physics at the University of Nebraska from 1927 to 1929. He also served as physicist and metallurgist for Baker & Co., Inc., Newark, N. J. In 1942 he joined the War Production Board as chief of precious metals and mercury section. Since 1945 he has been with the NRL.

Harold Margolin (p. 667) is a graduate of Yale University. Dr. Margolin is presently a research associate in the research div., New York University. Born in Hartford, Conn., he was employed by the Underwood Elliott Fisher Corp. after graduation. From 1944 to 1946 he was in the Navy and then joined the staff of NYU. This Member of AIME enjoys bridge and dabbles with making home movies.



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IN testing high-voltage circuit breakers, engineers at General Electric Company *intentionally* short-circuit two huge motor-driven generators. Each of these test generators is normally rated 125,000 kva, but provides short-circuit currents as high as 182,000 amp, instantaneous, offset wave, corresponding to about 1,625,000-kva rms symmetrical short-circuit duty. Such operation causes tremendous mechanical stresses to build up inside each machine. These stresses create a torque that tries to twist loose the 200-ton stator assembly. But no damage is done! These powerful machines are mounted on U·S·S CARILLOY steel plate springs that cushion the shock and then damp out any vibrations that follow.

Each generator is short-circuited repeatedly during tests, sometimes as often as 40 times an hour. And *each time*, the springs must absorb a terrific shock torsion load. They also must provide rigid vertical support for the stator. It takes a tough, very durable steel to stand up in this severe service.

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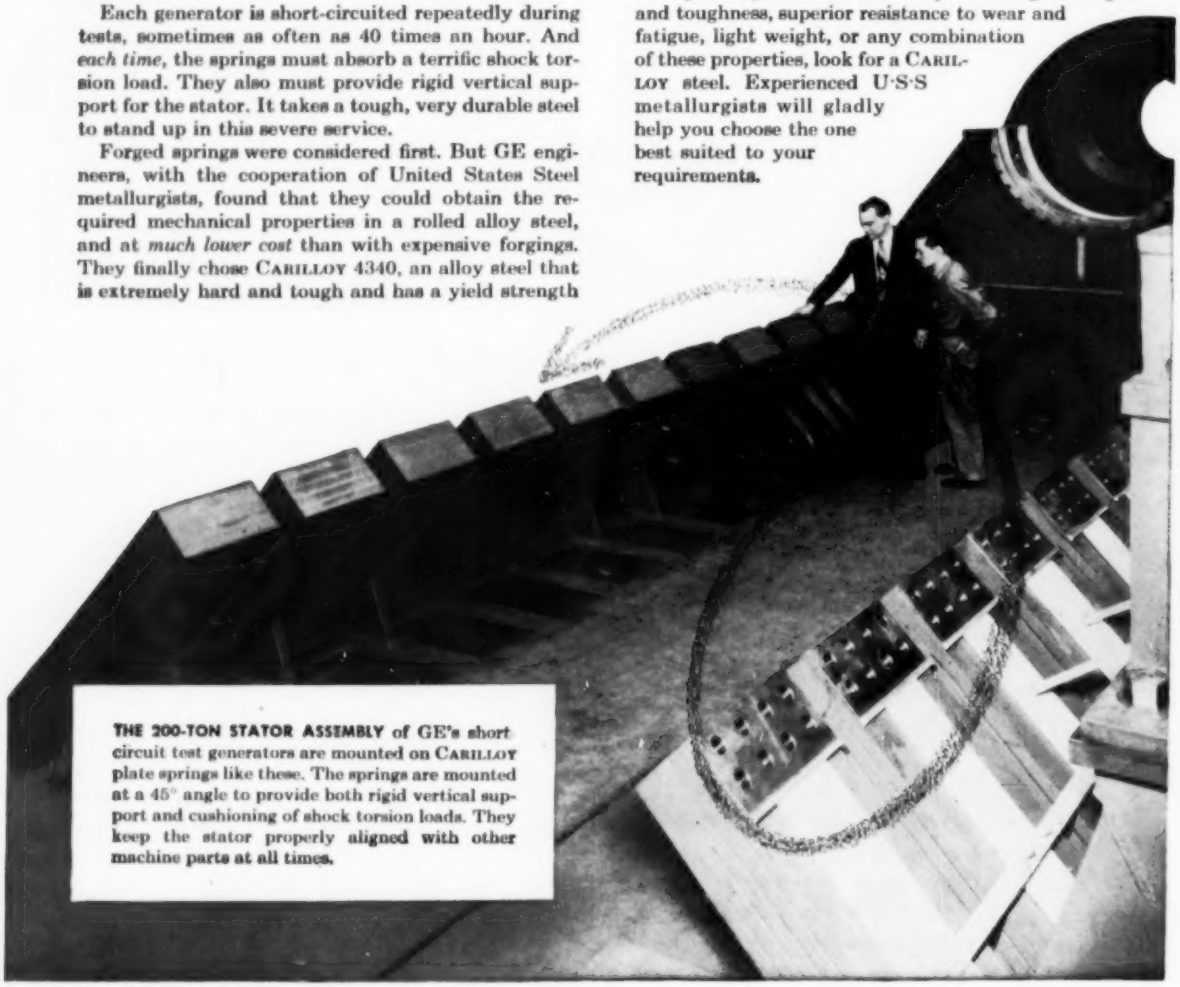
Also important was the fact that CARILLOY 4340 is easy to heat treat. Plates were furnished to GE quenched, tempered, and special cut and stress relieved.

This steel is giving excellent service. Initial vertical displacement of each generator was only 0.35 mils. Under the most severe short circuit, developing a whopping *8 million lb. ft.* of torque, frame rotation is about ½ in. each way at the point of attachment of the springs. The axial centerline of the machine stays within 30 mils of its normal position. These movements are sufficient to cushion the shock effectively.

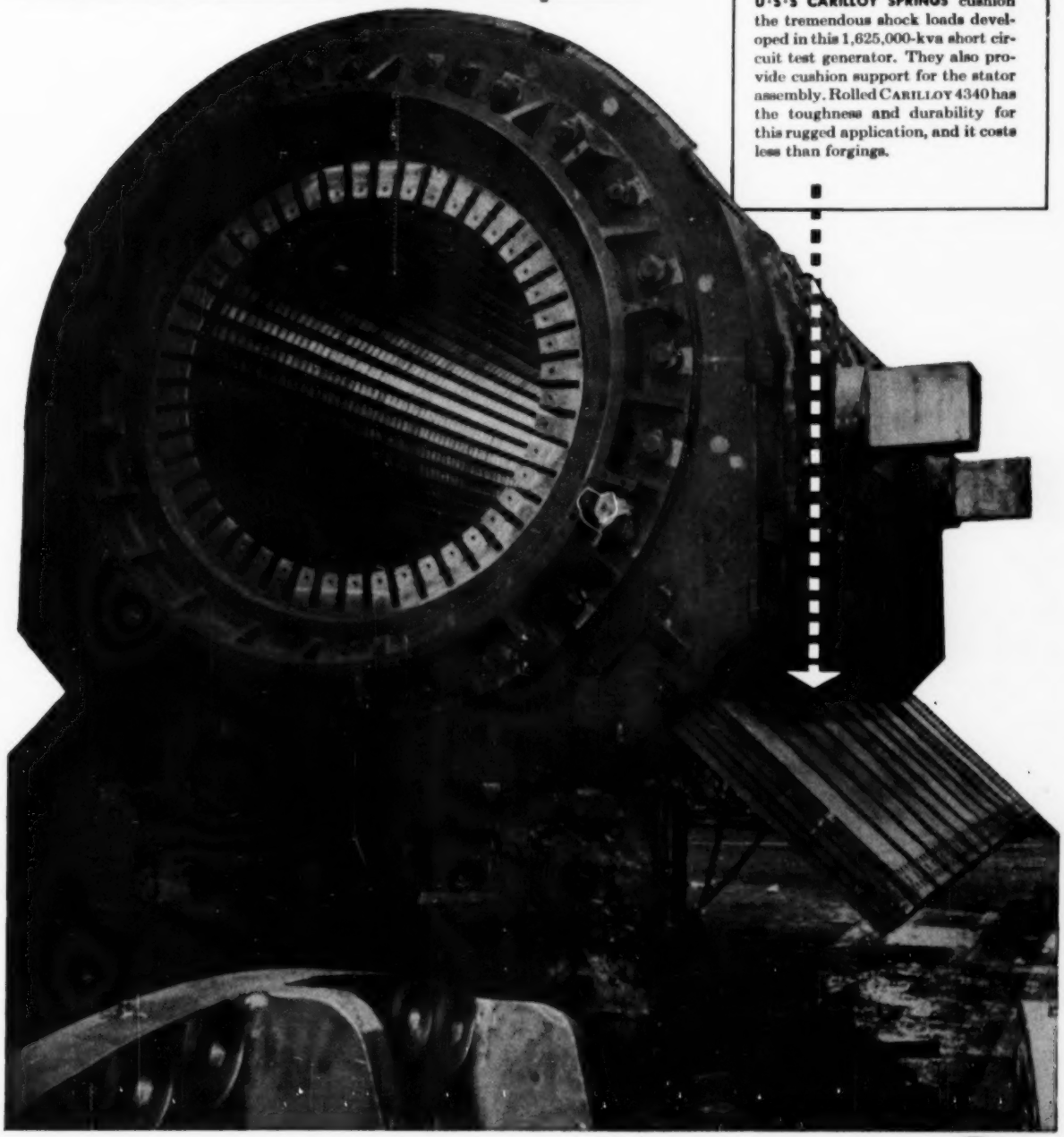
Any time you need a steel to provide high strength and toughness, superior resistance to wear and fatigue, light weight, or any combination of these properties, look for a CARILLOY steel. Experienced U·S·S metallurgists will gladly help you choose the one best suited to your requirements.



THE 200-TON STATOR ASSEMBLY of GE's short circuit test generators are mounted on CARILLOY plate springs like these. The springs are mounted at a 45° angle to provide both rigid vertical support and cushioning of shock torsion loads. They keep the stator properly aligned with other machine parts at all times.

8 million lb.-ft. torque!

U·S·S CARILLOY SPRINGS cushion the tremendous shock loads developed in this 1,625,000-kva short circuit test generator. They also provide cushion support for the stator assembly. Rolled CARILLOY 4340 has the toughness and durability for this rugged application, and it costs less than forgings.



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Recent Graduates for process industry equipment. (c) Ceramists and metallurgists for research dept. (d) Ceramists, metallurgists for manufacturing depts. Location, upstate New York. Y8058.

Metallurgical Engineer, metallurgical degree, with up to 5 years' experience desirable, to work with laboratory division. Salary, $4680 to $5400 a year. Location, upstate New York. Y8473(b).

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— Coming Events —

April-May, Empire Mining and Metallurgical Congress, Australia and New Zealand.

May 1-6, Purdue Eighth Industrial Waste Conference, Purdue University, Memorial Union Bldg., Lafayette, Ind.

May 11-14, American Mining Congress, coal show, Public Auditorium, Cleveland, Ohio.

May 14-16, AIME, Pacific Northwest Metals and Mineral Conference, joint meeting, Metals Branch and Industrial Minerals Div., Ben Franklin Hotel, Seattle.

May 14-23, International Petroleum Exposition, Tulsa.

May 15, AIME, Southeast Section, inspection trip, Southern Railroad, E. Norris yards, Irondale, Birmingham, Ala.

May 16, AIME, St. Louis Local Section, Presidential visit and Ladies' Night, St. Louis.

May 18-22, National Materials Handling Exposition, Convention Hall, Philadelphia.

May 20, AIME, National Open Hearth Steel Committee, Western Section, field trip.

May 22, AIME, National Open Hearth Steel Committee, Eastern Section, plant visit, Bethlehem Steel Co., Bethlehem, Pa.

May 24-28, Scientific Apparatus Makers Assn., annual meeting, Greenbrier, White Sulphur Springs, W. Va.

June 3-7, Chemical Institute of Canada, Windsor, Ont.

June 15-19, Basic Materials for Industry, conference, Hotel Roosevelt; exhibition, Grand Central Palace, New York.

June 16-19, American Welding Society, Shamrock Hotel, Hall of Exhibits, Houston.

June 16-Sept. 4, Gordon Research Conferences, AAAS, New Hampton School, New Hampton, N. H.

June 29-25, American Institute of Chemical Engineers, University of Michigan, Ann Arbor, Mich.

June 29-July 3, ASTM, annual meeting, Chalfonte-Haddon Hall, Atlantic City, N. J.

Sept. 11, AIME, St. Louis Local Section, Lead Belt Meeting, tour St. Joseph mines and mills.

Sep. 13-16, Electrochemical Chemical Society, Inc., Ocean Terrace Hotel, Wrightsville Beach, N. C.

Sept. 21-23, American Mining Congress, Metal and Nonmetallic Mineral Mining Convention, Olympic Hotel, Seattle.

Oct. 7-9, National Assn. of Consulting Engineers, south central region, Mayo Hotel, Tulsa, Okla.

Oct. 8-9, National Conference on Industrial Hydraulics, annual meeting, Hotel Sheraton, Chicago.

Oct. 9, AIME, National Open Hearth Steel Committee, Eastern Section, fall meeting, Warwick Hotel, Philadelphia.

Oct. 15, AIME, National Open Hearth Steel Committee, Southwestern Section, fall meeting, Baker Hotel, Dallas, Texas.

Oct. 19-21, AIME, Institute of Metals Div., fall meeting, Hotel Allerton, Cleveland.

Oct. 19-23, National Metal Congress and Exhibition, Cleveland.

Oct. 27, Assn. of Consulting Chemists & Chemical Engineers, 25th anniversary annual meeting, Hotel Belmont Plaza, New York.

Oct. 28-31, AIME, El Paso Fall meeting, El Paso.

Oct. 29-31, National Council of State Board of Engineering Examiners, Annual Meeting, Plaza Hotel, San Antonio, Texas.

Oct. 30-31, AIME, National Open Hearth Steel Committee, Southern Ohio Section, Deshler-Wallick Hotel, Columbus.

Nov. 6, AIME, National Open Hearth Steel Committee, Pittsburgh Section, off-the-record meeting, William Penn Hotel, Pittsburgh.

Dec. 13-16, American Institute of Chemical Engineers, annual meeting, Hotel Jefferson, St. Louis.

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SOLID STATE PHYSICISTS — Ph.D. or equivalent in experience in physics with a specialty in solid states work preferred. Will study electrical and optical behavior of semi-conducting materials.

METALLURGISTS — advanced degree or experience required. Will work on metallurgical preparations of semi-conducting devices.

ELECTRONIC — with graduate work or experience in product or circuit design and development.

Send complete resume to:

Mr. Robert L. Koller

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Electronics Division

WOBURN, MASS.

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Special Grade	50 to 55%	2%	0.50%
Open-Hearth Grade	50 to 55%	8%	3%
Foundry Grade	50 to 55%	approx. 10%	3%

Vanadium Oxide (3 Grades)	Composition		
	Vanadium Oxide (V_2O_5)	Sodium Oxide (Na_2O)	Calcium Oxide (CaO)
Fused	86 to 89%	approx. 10%	approx. 2%
Sodium Polyvanadate (Red Cake)	approx. 85%	approx. 9%	approx. 2%
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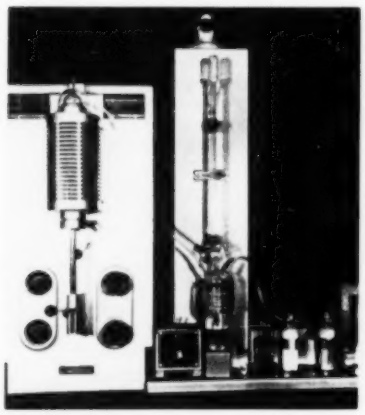
Students are Requested to Apply Directly to Manufacturers for Bulletins and Information

1—CARBON, SULPHUR DETERMINATION: A new unit that employs radio frequency inductive



heating tests for both carbon and sulphur in a single operation in 10 min, was introduced by *Lindberg Engineering Co.* The unit is capable of heating samples to temperatures above 3000°F.

2—BOX DRAW FURNACE: An electric box draw tempering furnace designed for applications requiring no protective atmosphere is available from the *Westinghouse Electric Corp.* This furnace is used for batch drawing or tempering high volume work that can be handled on trays. It is available in six sizes ranging from 15x24x12 in. to 42x72x30 in. The five larger sizes utilize 480 v and the smallest 240 v.

3—SURFACE COATING: Stainless steel coating type of finish is now available in colors including black, white, gray, ivory, green, red, orange, and yellow. Steelast can be sprayed, brushed or dipped and will withstand constant wet heat up to 180°F and can stand dry heat up to 300°F. It dries to handle in 30 to 60 min after spraying; 2 to 3 hr for recoating. It may be force dried in 10 min. *Steelcote Mfg. Co.*

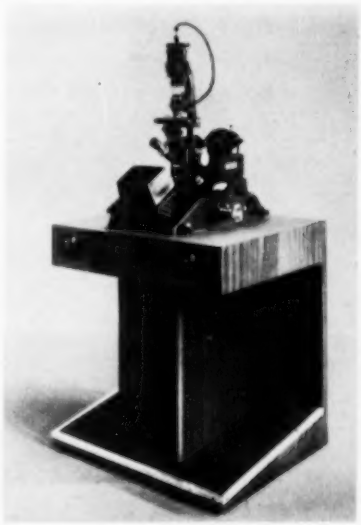
5—STONE CUTTING: A new process for cutting and piercing stone, cement, concrete, and refractory materials has been introduced by *Eutectic Welding Alloys Corp.* An arc rod is used upon nonconductive material and without a ground connection. The DynaTrode creates its own arc.

6—LADLE STOPPER: A new ladle stopper that will not crack from heat shock, offering almost absolute control over pouring of molten steel into ingot molds, is announced by *Electro Refractories & Abrasives Corp.* It is made of fire clay, graphite and a special synthetic ingredient. The product is said to suffer only slight change in contour due to erosion by steel.

7—CASTING SEALANT: Process developed by *Polyplastex International Corp., Inc.*, presents chemical properties that render the impregnated casting impervious to corrosive attack of noninflammable hydraulic fluids, in addition to the physical properties required to insure absolute elimination of microporosity. The sealant has also been found to provide a good base for a variety of coatings.

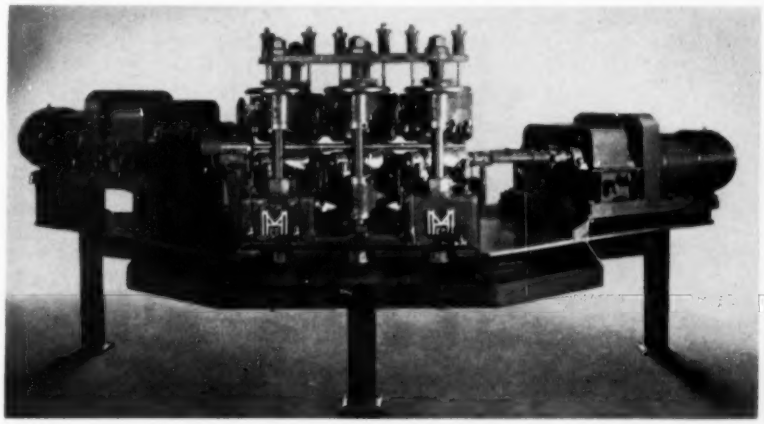
8—ELECTRONIC CONTROLLER: These controllers made in a wide selection of temperature ranges from 0-400°F to 0-3000°F, have been announced by *Bristol Co.* The new controllers actuate relays, electric contactors, solenoid valves, and motor valves to provide close temperature control of a wide variety of furnaces, ovens, kilns, salt pots, plastic molding machines, and other heating equipment.

10—UNIVERSAL MICROSCOPE: A universal microscope designed to do research or routine work in metallurgy, petrography, and biology with



speed, ease, and comfort has been announced by *Opplem Co.* Magnifications from 4X to 3000X can be run with continuity on the ground

9—ROTARY STRAIGHTENER: Several design improvements in the tubing and pipe rotary straightener have been announced by *Mackintosh-Hemphill Co.* The straightener, two 3-hp motors, and drive spindles are now mounted on a unit steel table. A circulating flood lubrication system for the straightener's six driven rolls is an integral part of the machine design. Model AX guideless rotary straightener continues to have the capacity to straighten tubing or pipe from ¼ to 1½ in. in diam. It has a production speed variable up to 350 ft per min.



glass screen of the camera. The instrument provides everything needed for macro and microphotography, polarization, phase, dark field, oblique, and standard microscopy in transmitted and reflected light.

11—DEGREASING AGENT: A finely ground cellulose fibre material with high absorbency is now available for use in degreasing and drying operations employing tumbling mills. Called Cottentex, it absorbs more than eight times its own weight. *Cottentex Mfg. Co.*

Free Literature

20 — SPECTROPHOTOMETERS: A 28-page catalog of *Beckman Instruments, Inc.* ultraviolet and visible spectrophotometers has been released. All sample cells and other accessories are included with detailed descriptions.

21—SPECIMEN GRINDING: Bulletin describes equipment available for fine grinding of specimens with emery papers or wet type silicon carbide papers or cloths. Both hand and motor powered grinders are included. *Buehler Ltd.*

22—ARCALOY ELECTRODES: The complete line of Arcaloy stainless steel electrodes is described in catalog released by *Alloy Rods Co.*

23—POLYESTER FILM: Up-to-date information on the physical, electrical, and chemical properties of Mylar polyester film and suggested applications is contained in bulletin issued by *E. I. Du Pont De Nemours & Co.* Charts, diagrams, and tables are included.

24—EXTRUDER: A new model of the Stokes-Windsor RC-100 extruder is described in bulletin No. 550 available from *F. J. Stokes Machine Co.* The advantages, capacity, specifications, and types of product this twin screw extruder produces are described. Sheet can be made in widths up to 53 in.; thicknesses can be as low as 8 mils.

25—INDUCTION MOTORS: A new brochure presenting the complete line of Lima Dripproof induction motors has been completed. Speed torque curves, frame number charts, dimensions and specifications for motors from 1/3 to 150 hp are given. Descriptions of variations for optional mounting and special purpose applications are also listed. *Lima Electric Motor Co.*

26—IDEA BOOK: *General Electric Co.* has announced a new bulletin entitled *Industrial Power Distribution Idea Book.* It is a guide for plant engineers on electric power distribution practices in large and small plants.

27—TEMPERATURE CONTROLLERS: Catalog ND47 contains complete data about moderate priced electronic instruments for two position or proportional control. *Leeds & Northrup Co.*

28 — METALLURGICAL POLISHING: A 20-page booklet describes and includes samples of emery, aluminum oxide and silicon carbide papers and 12 polishing cloths. Trade names, particle sizes, etc., are given for many abrasives. *Buehler Ltd.*

29—SILICONE RUBBER: The properties and processing of silicone rubber as an insulating material for wire and cable are explained in bulletin CDS-13 issued by *General Electric Co.*

30 — MILLIVOLTMETER INSTRUMENTS: Indicators, indicating electronic controllers and excess temperature safety cut off controllers are described in catalog 1053. *Minneapolis-Honeywell Regulator Co.*

31—SODIUM: Handling and use of dispersed sodium, bulk sodium and sodium sand are discussed in brochure issued by *Ethyl Corp.* Chemical and physical properties, availability in various forms, handling, and storing are also included.

32—CLAD STEEL: Guidebook *Cleaning and Maintenance of Clad Steel Equipment* has been published by *Lukens Steel Co.* Solution formulas for cleaning specific kinds of surfaces and cleaning procedures for types of surface contamination that are potentially harmful are included.

33—MICROFILM ENLARGEMENT: Folder on continuous microfilm enlargement service has been released by *Remington Rand, Inc.* Accurate reproductions produced at high speed are available with this new service. It allows a convenient means to restore any or all of a microfilm file.

34—POWER CONVERSION: The complete line of *General Electric Co.* metallic rectifier power conversion units is described in bulletin GEA-5658B. Design features, performance, circuits, dimensions and ratings for the complete line rated from 125 to 250 v dc are included.

35—AUTOMATIC SWITCHES: Fully automatic pneumatic tube system for transporting messages, small tools, laboratory samples or other small articles is explained in bulletin published by *Lamson Corp.*

36—INSPECTION EQUIPMENT: Production and single frequency comparators for inspecting ferrous material or parts are described in catalog. Dual frequency comparator for inspecting both ferrous and nonferrous metals is also discussed. *Magnetic Analysis Corp.*

37—AUTOMATIC WASHERS: A new line of compact, forced circulation washers in three sizes has been announced by *Ipsen Industries.* They are designed to speed metal cleaning through automatic cycling and by combining washing and rinsing in a single operation.

38—FORGING SYSTEM: A complete story of an actual continuous forging system is presented in pamphlet *Automatic Rotary Hearth Furnace Applied to Continuous Forging Systems.* Featured is the Hagan automatic rotary hearth furnace. This 30-ft diam, doughnut type furnace automatically heats sheared billets and supplies them on a continuous basis to the forging production line. *George J. Hagan Co.*

Journal of Metals
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May

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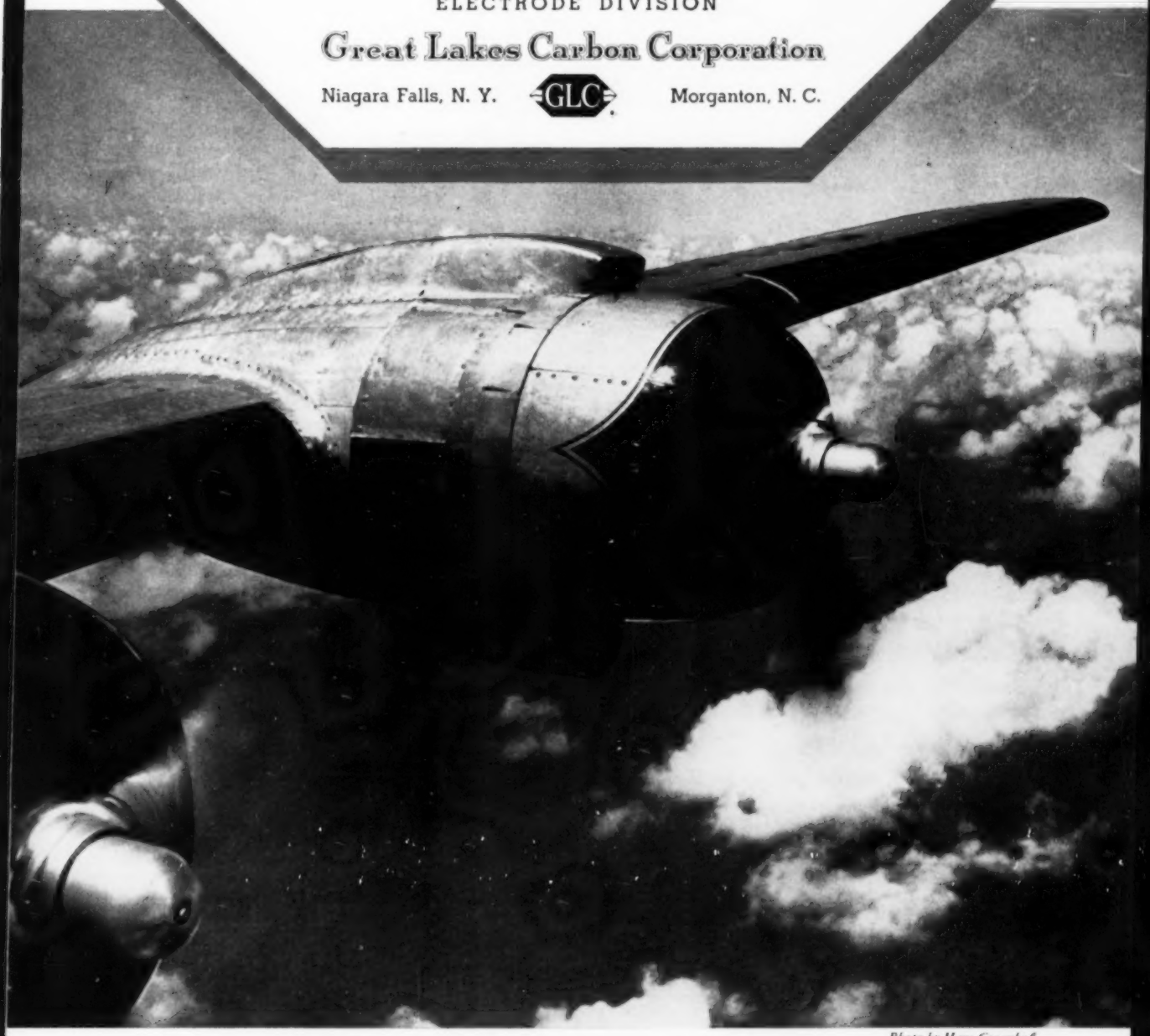

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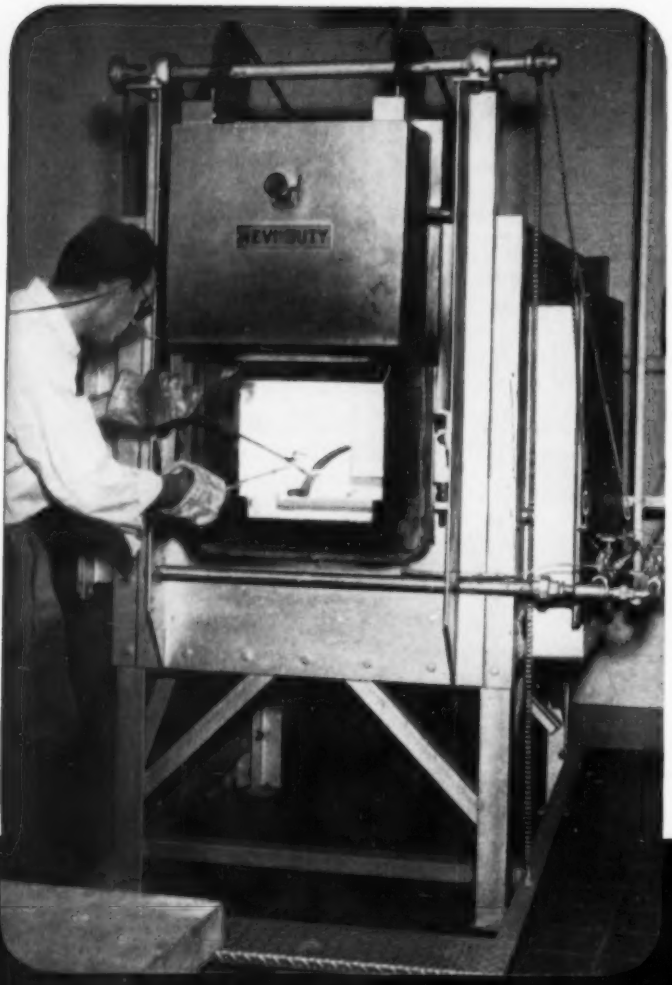

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A large automotive and ordnance manufacturer uses Hevi Duty high temperature furnaces in its metallurgical laboratory for determining production heat treating cycles.

These furnaces provide the uniform heat needed in this important work. Temperatures are accurately controlled to 2600° F. For day in day out production or precision laboratory work rugged Hevi Duty furnaces will meet your requirements. Write for bulletin IND-741 today.

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Heat Treating Furnaces... Electric Exclusively
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Industrial Notes . .

• A new alloy said to be tailor-made for electrical contact springs in large accounting and billing machines has been announced by Battelle Memorial Institute. The copper base alloy was developed as a possible replacement for beryllium-copper alloys—said to be in short supply. Electrical conductivity and strength are reported not to be as high in the new material as those of copper-beryllium alloys.

• The Atomic Energy Commission announced successful operation of a prototype of the nuclear reactor to be used to power the atomic submarine now under construction by General Dynamics Co., subsidiary of Electric Boat Co. The prototype is in operation at Arco, Idaho and will be used to study problems connected with eventual use of nuclear reaction powered undersea craft. Some predictions place the atomic submarine's speed at about 20 knots underwater. At Oak Ridge, the AEC reports that electrical energy in useful amounts has been produced from operation of a new type reactor. The reactor was brought up to its full design power of 1000 kw of heat output. Reactor steam was switched to a turbine-generator and about 150 kw of electricity produced.

• *Installation of special belt conveyors to aid unloading bauxite from a new type self unloading ore carrier has been completed by Rust Engineering Co., of Birmingham and Pittsburgh. The equipment was installed for Reynolds Metals Co. at the port of Mobile for transfer of ore from the ship to rail facilities. The new conveyor is in addition to a $750,000 bulk-handling conveyor system Rust completed last year for the Alabama State Docks and Terminals at Mobile. The new equipment will handle Jamaican bauxite from a special 12,500-ton ore carrier. The ore will go to the Reynolds' Little Rock, Ark. plant.*

• U. S. Steel Corp. is planning construction of a new research center for developing new and better steel and steel technology. The center will be located in Allegheny County, near Pittsburgh. Tentative specifications call for three buildings with a combined floor space of about 132,000 sq ft: A main laboratory building, an experimental engineering shop, and an industrial process development building. Initially, the center will be devoted to applied research and better coordination of U. S. Steel research.

• Mutual Security Agency is sponsoring a team representing the French pressed metals industry visiting facilities in the U. S. Operating under the technical assistance program, the group will observe manufacturing techniques, new processes and machinery, and labor-management relations in the U. S. industry. The team will visit operations working with hot and cold steel, as well as small cutting operations using ferrous and nonferrous sheet metal.

• Electro Refractories & Abrasives Corp. is preparing for full scale production of what is said to be a radically new type of ladle stopper. The firm reports that the new stopper will not crack from heat shock, which implies a major advance in steelmaking. Electro's stopper was perfected after 10 years of research and is made of fire clay, graphite, and a special synthetic ingredient.

• Experiments started by the Government to determine whether alumina could be made from clay-bearing ores near Salem, Ore., will be continued by the Harvey Machine Co. Harvey recently purchased the Government-owned alumina plant at Salem for a reported $325,000. A 10 pct down payment with the remainder to be paid in quarterly installments over 10 years were said to be the conditions of the sale.

• American Cyanamid Co. is operating a new Atomic Energy Commission laboratory at Winchester, Mass., whose primary aim is development and improvement of methods for recovery of uranium from its ore. Special attention will be given to recovery from low grade ores. The laboratory receives ore samples from areas throughout the world. Present staff numbers about 80, half with technical degrees.

• *U. S. Steel Corp. may take a 49 pct interest in a French-American company to be set up for exploration of rich manganese deposits in French West Africa. U. S. Steel will undertake to buy 300,000 tons of manganese yearly when production starts—about 1960. French Overseas Mining Office will have a 21 pct interest; Compagnie Miniere de L'Oubangui Oriental, 15 pct; and Mokta El Hadid, 15 pct.*

• Fifty nine coke ovens will be constructed by Willputte Coke Oven Div., Allied Chemical & Dye Corp., for the National Tube Div., of U. S. Steel Corp., at Lorain, Ohio. The ovens will have a capacity of 290,000 tons of coke per year for use as blast furnace fuel.

We'll build you a complete artillery shell forging plant and hand you the key

Salem-Brosius possesses a unique background of experience in the design and fabrication of shell forging plants which dates back well before World War II, and makes this organization the ideal source for such facilities. We are equipped to assume the entire contract, or any segment thereof. Salem-Brosius lays claim to being highly skilled in the design and construction of plants incorporating heating furnaces, descalers, preform presses, shell-forming mills, brooder furnaces, quench tanks, materials handling equipment, and all the hydraulic, pneumatic, electrical, and fuel systems and controls necessary to efficient and economical operation. If you are considering entry into the rapidly expanding business of producing ammunition to insure our country's safety, or contemplating expansion of your current operation, it will pay you to contact us. We'll build the complete plant and hand you the key or help you with any part of the problem. Write, wire or phone.

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Aluminum is *Why*

Single press stroke saves $10.00 worth of machining

Hogging this landing gear plunger out of bar stock wasted half the aluminum in chips—tied up valuable machine capacity—squandered costly man-hours.

Because even tough aluminum aircraft alloys are ductile and formable, Alcoa engineers and aircraft designers saw the savings possible by impact extruding this part. Today, a slug of tough aluminum alloy is placed in a shallow die—a punch strikes it—the aluminum squirts upward, faster than the eye can follow. Almost instantaneously a plunger is ready for finish machining and assembly—a saving of $10.00 per part.

ALCOA is *How*

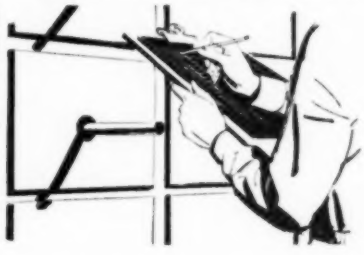
Though impact extruding is not new, it has been confined to simple, cup-shaped parts of workable alloys. But here, a *tough* alloy has been extruded. Nor is the shape simple. It is *thick bottomed, heavy walled* and the open end is flared, all an integral part of the extruding process. Impact extrusions, as Alcoa makes them, compete with almost every fabricating process. Let your local Alcoa Sales Engineer help you explore their possibilities. Aluminum Company of America, 879-E Alcoa Bldg., Pittsburgh 19, Pa.

Alcoa Aluminum

ALUMINUM COMPANY OF AMERICA

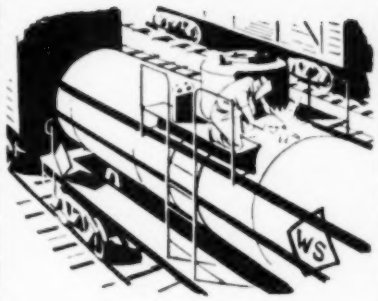
This strong, one-piece oil filter housing is an aluminum impact extrusion, produced by Alcoa in a single press operation. Formerly, this part required several operations.

Fabricating this double-wall fuse cap formerly involved an expensive welding operation. Now it is impact extruded by a single press stroke at Alcoa's Edgewater plant.

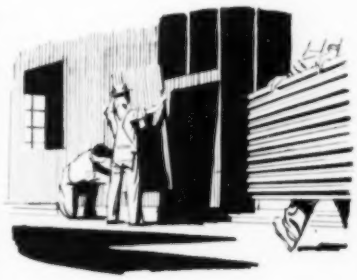


Used in electronic work, this impact-extruded can demonstrates an extreme length-to-diameter ratio which goes far beyond limits formerly believed possible.

Alcoa Aluminum takes all finishes that other metals will take—plus gleaming, rust-resistant anodic coatings which are best on aluminum.

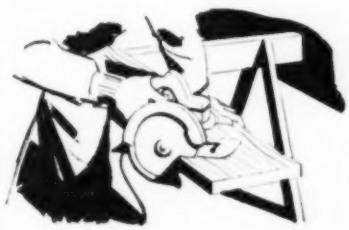


While large structures are usually arc-welded, aluminum assemblies also can be joined by torch welding, all three resistance-welding processes—spot, seam or flash, and brazing.



*
Lightweight, easy-to-handle Alcoa Industrial Building Sheet (only 56 pounds per square) goes up fast . . . reduces dead load . . . stays good looking without painting or finishing.

Corrosion-resistant bearings of Alcoa Aluminum lower diesel maintenance. They are solid metal all through.



Alcoa Aluminum Die Castings can weigh one-third as much as heavy metal castings, yet provide great strength. They are exceptionally easy to machine . . . take all finishes . . . often cost less.

*
Almost any shape can be produced as an Alcoa Aluminum Extrusion—hollow, semihollow, solid. Extrusions put the metal where it's most needed, yet use less metal than rolling or fabricating by welding or riveting.

Products marked* are available from your local Alcoa Distributor listed here

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Pittsburgh
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Houston
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Seattle
Pacific Metal Co.

WISCONSIN
Milwaukee
Steel Sales Corp.

Books for Engineers

Physical Chemistry of Metals, by Lawrence S. Darken, Robert W. Gurry, and Michael B. Bever. *McGraw-Hill Book Co.* $8.50, 535 pp., 1953.—A text for senior and graduate students, this book presents fundamental aspects of the physical edge of general physical chemistry, and metallurgically related substances to metallurgists with a minimum knowledge of general physical chemistry and to chemists with a minimum knowledge of metals. First section of the book is devoted mainly to descriptions of the solid and liquid states of alloys and metals from the viewpoint of the chemist rather than the solid state physicist. Classical chemical thermodynamics and the third law are dealt with in the second section. The former is developed from first principles and related directly to metallic and metallurgical examples.

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Atomic Theory for Students of Metallurgy, by William Hume-Rothery. *Institute of Metals.* $3.50, 331 pp., 1952.—The metallurgical student is often unfamiliar with the electron theory which is the general background of atomic theory. The book is written primarily for advanced students and research workers, and is intended to help in understanding of the underlying ideas. Following an introductory chapter, headings are: Structure of the free atom; assemblies of atoms; free electron theory of metals; Brillouin-zone theory of metals; electrons, atoms, metals and alloys. Properties of metals discussed are restricted to those for which the theories provide a reasonably straightforward explanation.

Radioisotopes in Industry, edited by John R. Bradford. *Reinhold Publishing Corp.* $8.00, 309 pp., 1953.—All important industrial uses of radioisotopes are covered in the book, based on a series of lectures presented at Case Institute of Technology. The original material has been revised to include the most recent material. Related topics, such as laboratory construction, shielding, personnel, protection, handling techniques, and radioactive waste disposal, are also investigated. Recently developed industrial uses for gross fission products are covered.

Precision Measurement in the Metal Working Industry. Prepared by the Dept. of Education of International Business Machines, Inc. *Syracuse University Press.* $6.50, 365 pp., 1952. —This revised manual was prepared for use in the factory training program of IBM. The volume opens with a general statement on measurement. Successive chapters deal with line-graduated instruments, precision gage blocks, plug, ring and snap gages, thread gages, dial gages and test indicators, micrometers, and verniers, surface plates and accessories, angle-measuring instruments, comparators, optical instruments and surface finish measurement, measuring machines, hardness testers, and non-destructive testing methods. Detailed drawings and photographs are widely used.

Statistical Yearbook, prepared by the Statistical Office of the United Nations. *Columbia University Press.* $7.50, 554 pp., 1953.—Contains as complete a collection as possible of world statistics with much of interest to metallurgists and mining engineers. Production, consumption, and industrial growth are presented in table form. Minerals and metals are treated individually and by nation.

ORDER YOUR BOOKS THROUGH AIME—Address Irene K. Sharp, Book Department. Ten per cent discount given whenever possible. Order Government publications direct from the agency concerned.

Minerals—A Key to Soviet Power, by Demitri B. Shimkin. *Harvard University Press.* $8.00, 452 pp., 1953. —The book is an economic evaluation of the position of the Soviet Union in respect to mineral wealth and development. Various sources are drawn upon, including the reports issued after completion of the various five year plans. The book is of considerable reference value despite the confusion which beclouds information concerning Soviet mineral progress. The theoretical study is the joint product of some of the staff of the Harvard University Russian Research Center. The author's approach to metals and nonmetals in the Soviet Union is systematic and he presents a great deal of data covering the period 1926 to 1950. Indicated weaknesses in the mineral economy are highlighted. Some considerable space is given to Soviet mineral future.

Mechanics of Materials, by Seibert Fairman and Chester S. Cutshall. *John Wiley & Sons, Inc.* $5.75, 420 pp., 1953.—An attempt is made to make the subject meaningful to students by showing how to apply theory to practical situations. The book contains numerous illustrative examples, completely solved, to aid the student in learning how to apply principles. A systematic treatment of combined stresses is also offered. A treatment of columns is designed to give the student a better understanding of the basic theory and its connection to the development of the empirical formulas used in practical design. Certain advanced topics have been eliminated.

Applied Inorganic Analysis, second edition, by W. F. Hillebrand and G. E. F. Lundell, revised by G. E. F. Lundell, H. A. Bright, J. I. Hoffman. *John Wiley & Sons, Inc.* $15.00, 1034 pp., 1952.—Aim of this second edition is to describe methods of separation and determination that have proved of value, or that promise to have value, in applied inorganic analysis, rather than to list or describe methods of limited application. New material has been added and some old has been discarded. Many of the chapters have been revised and some rewritten. Rewritten chapters are those on columbium, tantalum, tin, and the plati-

num metals. The chapter on uranium has been expanded in proportion to the wealth of new material on hand since the first edition. Brief discussions on the new topics, polarography, chromatography, flame photometry, fluorometry, mass spectrometry, X-ray analysis, are presented. Fundamentally unchanged are the chapters dealing with silicate and carbonate rock analysis. The book is intended to be supplementary to *Outlines of Methods of Chemical Analysis*, by Lundell and Hoffman.

Hydraulics and its Applications, fifth edition, by A. H. Gibson. *Constable & Co., Ltd.* $4.90, 813 pp., 1952.—The fifth edition of this textbook on the science of hydraulics and its applications to the design of hydraulic machinery has been revised to give an up-to-date picture of its subject. Written primarily with the needs of the student in view, the book may prove of value to those actively engaged in the practice and profession of hydraulic engineering. Knowledge of differential and integral calculus is not needed for a grasp of the greater part of the book.

Saudi Arabia, second edition, by K. S. Twitchell. *Princeton University Press.* $5.00, 231 pp., 1953.—The stated purpose of the book is to give the western world some idea of Saudi Arabia and its ruler King Abdul Aziz ibn-Saud. The second edition of the book brings the reader closer to changes which have taken place in that country between 1946 and 1952. An account of development of natural resources is presented, with chapters devoted to mining and petroleum advances. Geographical and geological data chapters are included in the first section. The book is illustrated with maps and photographs.

Low-Temperature Physics. Superintendent of Documents, *Government Printing Office.* $1.75, 291 pp., 1952. —The volume is composed of some 60 short papers dealing with some specialized aspect of low temperature physics. Subjects discussed are: Characteristics, properties, or behavior of an element or other substance; methods for the measurement of physical quantities or attributes at extremely low temperatures; and descriptions of research activities. A large proportion of the papers deal with superconductivity effects, including a few theoretical treatments of a general nature. In some instances only abstracts have been printed.

Essentials of Fluid Dynamics, by Ludwig Prandtl. *Hafner Publishing Co.* $6.00, 452 pp., 1952.—Mathematical treatment is held to a minimum in this translation of a recent German treatise on the flow of fluids. It is intended to provide a comprehensive and easily understandable account of the subject. Major sections are: Equilibrium of liquids and gases; kinematics and the dynamics of frictionless fluids; the motion of viscous fluids, turbulence, etc.; flow with appreciable volume changes (dynamics of gases); and miscellaneous topics. Many practical applications to hydraulics, aeronautics, and other subjects are included.

High Speed Photography, by George A. Jones. *John Wiley & Sons, Inc.* $6.50, 311 pp., 1952.—This is a comprehensive presentation of broad fundamental theory, practical operative techniques, and technical applications. Detailed descriptions are presented of lighting and camera equipment needed for still and motion pictures. A wide range of scientific, industrial, and commercial applications is covered with special requirements and limitations for each case. References and illustrations are widely used.

Physical Chemistry of Copper Smelting, by R. W. Ruddle. *Institution of Mining & Metallurgy.* About $3.00, 156 pp., 1953.—This short monograph on copper smelting from a physical-chemical viewpoint serves as a bridge between industrial practice and current research, and should be of considerable value to the smelter man, teacher, and researcher. It consists of an extensive review and correlation of the physical chemical knowledge relating to the constitution of copper matte, copper smelting slags, formation of magnetite, copper losses in slags, recovery of sulphur from smelter gases, and elimination of impurities during smelting. Also included are tables of thermodynamic data, a review chapter on chemical thermodynamics, and an outline of current practice in copper smelting. This is about the most comprehensive survey and review of the subject that has ever appeared.

Physical Chemistry of Melts. *Institution of Mining & Metallurgy.* About $2.15, 106 pp.—This volume contains the papers and discussion that formed the Symposium on Molten Slags and Salts held by the Nuffield Research Group in Extractive Metallurgy on Feb. 20, 1952, in London. Included are six papers on recent European research on slag constitution. This book should be a standard work for all researchers on slag constitution. It points out (as did the 1948 Faraday discussion on Physical Chemistry of Process Metallurgy) that European research in extractive metallurgy is active, enlightened, and in advance of U. S. research in several aspects.

Elements of Nuclear Reactor Theory, by Samuel Glasstone and Milton C. Edlund. *D. Van Nostrand Co., Inc.,* $4.80, 416 pp—This volume presents the whole field of nuclear reactor theory and provides the engineer with the basic knowledge for an understanding of reactor technology. The book, based on lectures given at Oak Ridge, begins with the more elementary physical concepts and processes of nuclear reactions, and then develops the fundamental principles involved in the calculation of the critical conditions for chain reacting systems — including both homogeneous and heterogeneous reactors — giving detailed mathematical derivations.

Electrical Measuring Instruments — Part I: General Principles and Electrical Indicating Instruments, by C. V. Drysdale and A. C. Jolley. Revised by G. F. Tagg. *John Wiley & Sons, Inc.* $12.50, 598 pp., 1952.—Beginning with general electrical principles and typical basic systems, this standard work continues with the mechanical design and construction of instrument mechanisms and the elements of electrical theory and design. Permanent magnet moving coil instruments, soft iron instruments, dynamometer ammeters, volt meters, watt meters, hot wire instruments, and electrostatic instruments are dealt with in detail.

Industrial Brazing, by H. R. Brooker and E. V. Beatson. *Iliffe & Sons Ltd.* $7.50, 344 pp.—This authoritative study of brazing covers in considerable detail all modern brazing methods—torch, furnace, high frequency induction, resistance, salt bath, and dip. Special considerations necessary in work on aluminum, stainless steels, beryllium copper, cemented carbides, and vacuum tube construction are dealt with separately. In addition, a general introduction to the processes and equipment employed is followed by a review of brazing materials and the most complete summation of the known factors governing joint design. One chapter discusses problems of selecting the most appropriate brazing process for various types of work and makes a useful contribution on methods of inspection and testing.

Fatigue of Metals, by R. Cazaud, translated by A. J. Fenner. *Chapman & Hall Ltd.* About $10.00, 334 pp.—The test data presented cover an unusually wide range of metals and alloys, and constitute a valuable reference. The material is taken from Dr. Cazaud's own investigations. Part of the book presents a survey of the general characteristics of the fatigue of metals under the separate or combined influences of stress, temperature, and immediate environment. The remainder of the book is devoted to the vitally important practical aspect of fatigue. An extensive up-to-date review of fact, practice, and experience is presented, which if applied should be helpful towards the improvement of design, selection of material, and use of more suitable workshop methods and finishes, as well as guidance at the inspection stage.

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Mining Abstracts

During recent months, several articles have been published in MINING ENGINEERING, the AIME Mining Branch publication, that bear on the activities of metallurgical operations. These articles are listed below for the convenience of the reader.

Copies of issues in which individual articles appeared can be obtained from AIME at the single copy price of the magazine, or photostats of the articles can be obtained at the regular prices for such work.

All articles listed have appeared during the latter half of 1952 and first three months of 1953.

June: **Jacksonville Plant Produces Titanium From Beach Deposit.** J. C. Detweiler. Recovery of rutile, ilmenite, zircon, and monazite from beach deposits.

Impact Crushing for Reduction of Hard-Abrasive Ores. W. W. West.

July: **Foreign Minerals and American Capital.** H. Dewitt Smith.

August: **Measurement and Evaluation of the Rate of Flotation as a Function of Particle Size.** T. M. Morris.

September: **Deming Mill—A Materials Handling Problem Solved.** Norman Weiss and H. W. Kaanta.

October: **Factors in the Economics of Heat-Treated Taconites.** Will Mitchell, C. L. Sollenberger, F. F. Miskell. Heat treatment of ore prior to comminution reduces power requirements for grinding, reduces grinding media wear, and improves recovery of iron values from a typical Minnesota magnetic taconite. Test data demonstrating this, as well as an analysis of the economics of commercial application of the technique are presented.

Periclase Refractories in Rotary Kilns. Leslie W. Austin.

November: **Microstructures in Iron Ore Pellets.** S. R. B. Cooke and T. E. Ban. The paper discusses the mineralogy, structure, and strength of magnetite pellets fired in air and in a neutral atmosphere at various temperatures. The information obtained from this investigation is used to explain the origin of the structures occurring in pellets produced by pilot plant operation.

December: **Chuquicamata Sulphide Plant.**

Magnetic Roasting of Lean Ores. Fred D. DeVaney. By means of a roasting technique quite different from older methods, it has been demonstrated that iron-bearing materials can be roasted as effectively as by any previously known, and at a much lower cost.

Effects of Alkalinity on the Flotation of Lead Minerals. Marston G. Fleming. General theories of alkali depression are disproved by this paper, which shows that its operation depends upon the nature of the mineral-collector system involved. Three mechanisms of alkali depression are examined and limitations of critical pH as a fundamental factor in flotation are discussed.

January 1953: **Storke Level—Key to $25 Million Climax Project.** Charles M. Cooley. The expansion program at Climax Molybdenum to be operating by 1954 includes a new mining level, crushing plant, 4000 ft conveyor, milling units, byproducts plant, and housing for 198 families. Completion of this program and low grade plant will add 10,000 tons per day to present capacity and several million tons to ore reserves.

Sponge Iron at Anaconda. Frederick F. Frick. Abstracted in JOURNAL OF METALS, March 1953.

Principles of Present Day Dust Collectors and their Application to Mining and Metallurgical Industries. J. M. Kane and R. H. Walpole.

February: **Annual Review Number for 1952 Mineral Resources.** Donald H. McLaughlin.

Big New Plants Highlight Beneficiation Development. E. H. Crabtree. Research on the iron ranges begins to bear fruit in big plant construction — Big coppers raise capacity, build at New Arizona, Michigan, Nevada properties — Hydrometallurgy comes to fore with new uranium ore treatment and the Chemico process for nonferrous metals.

March: **Electrical and Metallurgical Improvements at Kennecott's Utah Copper Div. Mills.** R. J. Corfield and A. G. Johnson. Modernization of the entire electrical system and improvement of flotation process efficiency is the twofold goal of the improvement program underway at the Arthur and Magna concentrators of the Utah Copper Div., Kennecott Copper Corp.

Operating Behavior of Liquid-Solid Cyclones. E. B. Fitch and E. C. Johnson. The operating behavior of liquid-solid cyclones is outlined, together with the nature and range of the process results obtainable, to serve as a background for engineers wishing to consider application of this process tool.

Sampling and Testing of Sinter. R. L. Stephenson and D. J. Carney. A sampling technique has been developed for procuring a sample of sinter representative of the entire depth of the sintering bed. The sampling method involves the use of an open-bottom basket that rides on the grate of the sintering machine and when removed contains a sample of the sintered product. Additional data have been obtained to indicate that the tumbler test is a suitable means of measuring sinter strength.

Drift of Things

TOO early spring may be the explanation for the poor turnout of mining and metallurgical engineer candidates at a large New York guidance conference, but it would be wiser to take a more serious view of this occurrence. Held a few weeks ago, the meeting was preceded by weeks of preparation and publicity in which many engineering societies participated. The meeting was the First Greater New York University downtown campus for splitting up anticipated to draw about 20,000 high school students from the New York area and environs. Seven of the theatres in the Paramount chain were available to hold the audience for a panel program on career opportunities which was to be televised and piped into the theatres. Following the morning theatre session, students were to adjourn to the New York University downtown campus for splitting up into workshop sessions devoted to seventeen different branches of the profession.

The day of the meeting dawned sunny, clear, and unseasonably warm. Attendance in the morning reached 2500, about one eighth of expectations, and dwindled to about 1100 in the afternoon as the good weather took its toll. Paul Lange, who was counseling mining, told to prepare for 100, found four students at his meeting room. George Kehl had seven at the metallurgy workshop of whom it appeared three were in the wrong pew being chemical engineering aspirants. This seems like pretty lean pickings for the mineral industries when the electrical, chemical, and mechanical groups each drew hundreds.

Mineral industry people connected with the conference were disturbed by this outcome of the meeting. They pose the question of whether or not this condition pertains throughout the country. From where we sit considerable evidence has come to hand to indicate that an apathy does exist among the nation's young people toward the mineral industries with the possible exception of petroleum.

Metropolitan areas represent our largest reserves of potential engineer-power. But here industry is obscured by being located in parts of town where school children seldom go or in such a haze of industrial grime as to be unattractive. With this bleak facade, which is usually associated with engineering, coupled to public ignorance of what engineers are, the profession is bound to suffer in engineer recruitment.

Our industry is the least known and has lost out in the glamour contest to electronics, nuclear science, rocket engineering and even industrial engineering. We excepted petroleum earlier because it evidently still has an aura of quick riches.

No outpost should be deserted by the simple thought that you wouldn't expect to find people interested in mining and metals in the New York area. They aren't interested because they don't know and it would be suicide to sit back and wait for the young people from the mining and smelter towns to fill the ranks.

In the March *Drift* we described what some are doing to publicize engineering and as a starting point for those who think the situation serious enough, we suggest an extension of these programs.

LAST September MINING ENGINEERING carried a news item about a $3.8 million loan by Defense Materials Procurement Agency to Westmoreland Manganese Corp. for mining and concentrating manganese deposits near Batesville, Ark. Westmoreland was to increase annual domestic production of manganese by 52,800 tons by concentrating low grade ores by the sink-float process at the rate of 6000 tons of raw feed per day to produce 240 tons of high grade manganese. The company put up $775,000 of its own money.

A recent clipping from a Batesville paper indicates that the operation is shut down before completion of the plant or opening the mine, the company having exhausted its funds. According to DMPA spokesmen that agency is not disposed to advancing more money.

Apparently this outcome might have been anticipated from the start if DMPA had looked closely enough at the project before okaying the loan. Before the end of September we had received a copy of a letter from Charles S. Blair of Birmingham, Ala., to Jesse Larson, administrator, DMPA. Mr. Blair wrote: "... Based on a somewhat intimate personal acquaintance with the Batesville, Ark., district and its manganese resources from World War I days, we are at a loss to understand the basis for such a tremendous loan to a relatively new and inexperienced company in this old and greatly depleted manganese field.

"... The output of high grade ore from the Batesville field has always been limited reaching only 7000 to 8000 tons during 1917 and 1918 and in about the same amount in 1944 and 1945 with the output declining to almost zero during the intervening years. The output in 1948 was recorded as 212 tons.

"... The high grade ores occur in residual clays, usually close to or even at the surface, and the entire field has been closely prospected by thousands of test pits and surface workings from which the ore production up to date has come, the miner screening out and recovering the ores from the clays in the same manner as the individual farmer does his potatoes from the garden.

"... This field has always included a large number of both purely speculative and important mining companies, some well financed, who have designed and built many various types of mechanical concentration plants, all of which have been entire failures due principally to the small tonnage available at any one locality.

"... As a matter of personal interest, I have asked Dun & Bradstreet for a financial report on the Westmoreland Manganese Corp. from which I quote:

'Holdings are stated by management to consist of 3000 acres of ore-bearing land owned outright with an additional 1450 under option for lease. Mine and process manganese ore. Equipment consists of one bulldozer and miscellaneous small mining equipment.'"

DMPA says it is not even considering putting any more money into the project unless the company can "show a better performance, including getting some money of its own."

John V. Beall

AFTER ten years of experimentation, Aluminum Co. of America has succeeded in heating a 7700-lb aluminum ingot to working temperatures through the use of ordinary household current. With the low frequency method, an ingot can be prepared 15 times faster than in a conventional furnace. Formerly requiring 12 hr, an aluminum ingot can be heated to 750°F in about 50 min. Essentially the new method consists of placing an unheated ingot in a specially designed electric coil. Sixty cycle current is applied to the coil. A heavy current is induced in the ingot itself by the current flowing through the coil, producing the temperature rise in the metal.

When the experiments were initiated, low frequency coils were considered impractical and inefficient. Only current of higher than commercial frequency was used, requiring expensive conversion equipment. Alcoa's first attempts were with 9600-cycle coils. As experiments continued, 1000-cycle, 360, 180, and finally 60-cycle coils were used. Alcoa reports that low cycle induction heating compares favorably with installation and operating costs of conventional equipment. Smaller equipment is permissible and induction heating provides greater flexibility in operation of big forging or extrusion equipment. Donald I. Bohn, Alcoa's chief electrical engineer stated that the new approach may prove invaluable in heavy press production of big aircraft aluminum forgings and extrusions.

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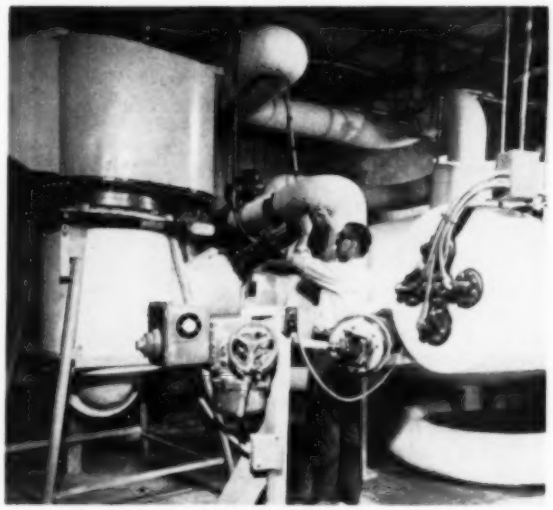
WHILE sodium is not a structural metal, it has a diversity of interest for the metallurgist. Its physical and chemical qualities are developing into a valuable addition to the tools available to the metallurgical engineer. Primarily it has found its widest application in hydride descaling, but it offers vast possibilities in the reduction of metal salts, a scavenging agent for metals, and a heat transfer medium. Used before the turn of the century for reduction of $TiCl_4$ to metallic titanium, research in this area of use is continuing today.

For all practical purposes it is tied with lead and zinc as the second cheapest metal, only iron is less expensive. Currently being produced at a rate of more than 200 million lb per year, it has long been produced as an intermediate in the manufacture of tetraethyllead as an antiknock compound for gasolines.

Sodium has the advantage of a low melting point (208°F), high boiling point (1628°F), high thermal conductivity, high heat capacity, low viscosity, low density, and the ability to wet iron and steel surfaces. Sodium dissolves, with heat evolution, in all proportions of molten lead, tin, antimony, and mercury. It is soluble to the extent of 2 pct by weight in molten zinc. All of these alloy products are quite brittle. Sodium will not alloy with iron and does not corrode steel at temperatures up to 800°F.

Because of the possible contact between water and hot magnesium in the die casting of magnesium a substitute for water cooling in cores has been found in the form of sodium. Sodium metal was used to partially fill these cores which moved from the vertical to the horizontal position during the casting cycle. Rapid heat transfer was obtained from top to bottom of the cores, prolonging core life, and allowing more rapid die casting machine operation. Sodium has also been used as a heat transfer medium in sodium cooled valves for aircraft engines. Sodium indirectly aids the metallurgist in this particular application by reducing operating temperatures and thus prolonging life.



Pump on the left circulates secondary liquid metal coolant in this experimental breeder reactor installation at Atomic Energy Commission's Idaho National Reactor Testing Station. This non-radioactive system can be approached by the operator. This is the type of operation in which sodium can be used as the secondary liquid metal coolant. White horizontal cylinder on the right is a heater for melting liquid metal upon occasional solidification when reactor is not operating.

About 2.5 million lb of sodium are used annually in hydride descaling. Sodium hydride descaling is applicable to any metal which does not react with molten caustic soda and whose physical properties are not adversely affected at the bath operating temperature of 700°F. It has found wide use in descaling titanium and hafnium. Metal to be descaled is immersed in a bath of molten caustic soda containing 1.5 to 2.0 pct sodium hydride. Sodium hydride is produced in the bath by reacting sodium metal with hydrogen gas in generator boxes along the sides of the bath. Oxide scale is reduced to metal or to lower oxides. Some of the reduced scale drops to the bottom of the bath while the remainder is removed when the parts are immersed in a water quench tank after removal from the hydride descaling tank. Treating time usually runs between ½ to 20 min.

Preliminary results on the use of sodium as a rectifier for oxidized chloride salt baths indicate that over-alkalinity resulted with consequent decarburization of the steel being treated. However, small amounts of sodium did accomplish deoxidation, without formation of any sludge. Experimental work is also progressing on the use of sodium as a deoxidizer and desulphurizer for molten steel.

Trends

AMERICAN Smelting & Refining Co. unveiled its new laboratory and concurrently held installation ceremonies for the Asarco Branch of the Science Research Society of America. Among those who took part in the dedication ceremonies were William Shockley, Bell Telephone Laboratories; Karl T. Compton, chairman of the Scientific Research Society of America and of the corporation of the Massachusetts Institute of Technology; Kenneth C. Brownell, president of American Smelting & Refining Co., and Albert J. Phillips, director of research. Asarco's laboratory is the first metallurgical research center to receive a charter in RESA. G. D. Stendahl is chapter president; H. E. Howe, vice-president, S. C. Carapella, Jr., secretary-treasurer.

The new laboratory is a mighty thing of pride to both Asarco management and employees. The long low, E shaped building is near South Plainfield, N. J., and in a large sense is symbolic of the growth of AS&R. During the company's formative years, individual plants carried on decentralized research concerned with their own problems. However, by 1925 the complicated nature of many of the problems indicated a definite need for a central laboratory. The new laboratory was established in a small portion of the Perth Amboy plant. Although metallurgical problems were of primary concern, the work of the group soon extended into the field of physical metallurgy. The unusually large number of metals handled by AS&R, new methods and processes, and services supplied to individual plants and customers dictated the construction of a new laboratory with room for future expansion.

The new location makes allowance for growth without interfering with the operation of the laboratory. The building is 210 ft across the front and each of three wings is 215 ft in length. The pilot plant, in the rear, is 70x140 ft with an overhead clearance of 30 ft. The laboratories are of one story construction, permitting access to the roof for ventilation. With the exception of the pilot plant and shop areas, a complete basement runs under the building. Piping and electrical conduits in the basement are exposed for easy alteration and repairs.

There are 65 individual laboratories and about 55 offices, shops, and storage rooms. There is also a library, drafting room, and pine paneled meeting room. Each office and laboratory is equipped with a thermostat for controlling temperatures no matter what the outside climate is. Hoods in laboratory areas remove exhaust air and a forced draft ventilating system supplies compensating air.

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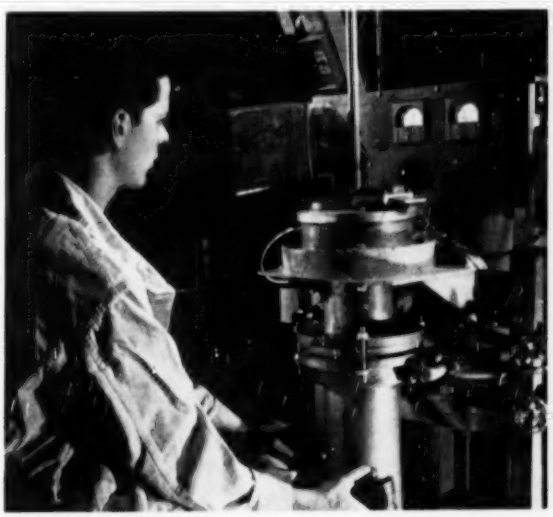
ZIRCONIUM, rated right behind uranium in importance in the construction of the long heralded atomic submarine, will be produced on a "full commercial scale," by Carborundum Metals Corp., according to Westinghouse Electric Corp. atomic scientists. In 1949 two-lb pieces of zirconium were experimented with under laboratory conditions. Today, industry is forming, rolling and machining parts from ingots 1-ft diam and weighing up to 500 lb. Considerable pilot plant work was done in 1951 and 1952 and with increased Atomic Energy Commission requirements the Bureau of Mines began production of the first commercial size ingots. The AEC recently announced that it has signed a contract with Carborundum for a 5-year order of zirconium sponge at less than $15 per lb. At the beginning of the Westinghouse submarine reactor project, zirconium sold at $250 per lb.

Raising the top at the end of the deposition process at the Westinghouse Atomic Power Div., plant discloses gleaming zirconium crystal bars emerging from the tank which has been flooded with water as a fire preventative. What were once thin wires now are solid bars of 99.9 pct pure zirconium. Metal is deposited in an irregular six-sided pattern.



Pieces of zirconium crystal bar are melted to form zirconium ingots in this electric arc furnace. Position of the arc is controlled by the operator by moving the magnet in his right hand. A mirrored view through a periscope tells him what is going on in the furnace. Furnace is operated at Westinghouse Atomic Power Div. plant.

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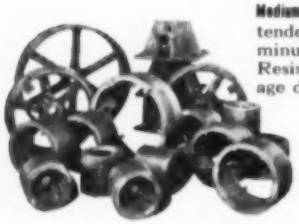

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Bush & Hamburg Streets
Baltimore 3, Maryland

Kutztown Foundry & Machine Corp.
Kutztown, Pa.

Montague Machine Company
Turners Falls, Massachusetts

Olney Foundry, Link-Belt Co.
180 W. Duncannon Avenue
Philadelphia 20, Pa.

The Richmond Foundry & Mfg. Co., Inc.
Hermitage Road
Richmond 20, Virginia

The Taylor & Fenn Co.
Saville Street
Windsor, Connecticut

Weatherly Foundry & Mfg. Co.
Weatherly, Pa.

MIDDLE SECTION...

American Brake Shoe Co.
Electro-Alloys Division
1936 Taylor Street
Elyria, Ohio

American Cast Iron Pipe Co.
P. O. Box 2603
Birmingham 2, Alabama

Baker Perkins, Inc.
1000 Hess Street
Saginaw, Michigan

Carondelet Foundry Co.
2101 S. Kingshighway
St. Louis 10, Mo.

Chicago Hardware Foundry Co.
2500 Commonwealth Avenue
North Chicago, Illinois

DeZurik Shower Co.
Foundry Division
Sartell, Minnesota

Duluth Brass Works Co.
5002 Ramsey Street
Duluth, Minnesota

The Duriron Co., Inc.
Dayton 1, Ohio

Engineering Castings, Inc.
405 South Linden
Marshall, Michigan

The Fahralloy Company
150th St. & Lexington Ave.
Harvey, Illinois

Frank Foundries Corp.
2020 Third Avenue
Moline, Illinois

The Hamilton Foundry & Machine Co.
1551 Lincoln Avenue
Hamilton, Ohio

Kingsport Foundry & Mfg. Corp.
East Sullivan and Main Streets
Kingsport, Tennessee

Michiana Products Corp.
P. O. Box 302
Michigan City, Indiana

Michigan Steel Casting Co.
1999 Guoin Street
Detroit 7, Michigan

Moran Flexible Joint Co., Inc.
215 West Main Street
Louisville 2, Kentucky

Reda Pump Co.
Bartlesville, Oklahoma

Ross-Meehan Foundries
1601 Carter Street
Chattanooga 1, Tennessee

St. Paul Foundry & Mfg. Co.
500 Como Avenue
St. Paul 3, Minnesota

Service Foundry Division
Avondale Marine Ways, Inc.
416 Erato Street
New Orleans 13, Louisiana

Shenango-Penn Mold Co.
Dover, Ohio

Standard Brass & Mfg. Co.
P. O. Box 1469
Port Arthur, Texas

The Sterling Foundry Co.
Wellington, Ohio

Texaloy Foundry Co.
1407 Hoefgen Avenue
San Antonio 3, Texas

The W. S. Tyler Co.
3615 Superior Avenue
Cleveland 14, Ohio

Wells Manufacturing Co.
7800 North Austin Avenue
Skokie, Illinois

WESTERN SECTION...

Centrifugal Casting Co.
3245 Cherry Avenue
Long Beach 7, California

Dameron Metal Sales Co.
927 South Sante Fe Avenue
Compton, California

Electric Steel Foundry Co.
2141 North West 25th Avenue
Portland 10, Oregon

Pacific Foundry Co., Ltd.
3100 19th Street
San Francisco 10, California

Stanley Foundries
6009 Sante Fe Avenue
Huntington Park, California

Utility Steel Foundry
3320 East Slauson Avenue
Vernon (L. A.) California

INCO

THE INTERNATIONAL NICKEL COMPANY, INC.

67 WALL STREET
NEW YORK 5, N. Y.

A $26 million plant turning out 6000 tons of titanium sponge has been proposed to the Government by the Crane Co., manufacturers of plumbing supplies. If the contract is signed and the loan forthcoming, Crane will be the largest titanium producer in the country, according to current standards. Partial production will be underway within 12 to 18 months if an agreement is reached, according to the company.

Steel strip measuring 0.00009 in. thick is being rolled at Allegheny Ludlum Steel Corp.'s subsidiary, Arnold Engineering Co., Marengo, Ill. The steel is rolled on a miniature cold mill developed by Armzen Co., Waterbury, Conn. Rolls are about the size of a half used pencil. The ultra-thin steel is especially useful in electronics because of the speed at which it becomes electromagnetized. However, the steel strip is extremely fragile, requiring great care in winding and finishing tape cores.

A compound of aluminum and antimony may compete with germanium and silicon for use in transistors, rectifiers, and other electronic devices. It is one of several developments which may arise from compounds of aluminum, gallium, and indium with arsenic and antimony. Studies sponsored by the Bradley Mining Co. at Battelle Memorial Institute have demonstrated that aluminum-antimony has electrical properties "at least as interesting as those of germanium and silicon."

New 46-in. blooming mill of Granite City Steel Co., is in operation. The mill, with its auxiliary facilities accounts for about 40 pct of the firm's $68 million expansion program which began in 1951. The program, expected to be completed this summer, will double Granite City's capacity. The mill is powered by two 5000-hp twin-drive electric motors.

Exclusive licensing has been issued to Pittsburgh Lectromelt Corp. to produce electric induction furnaces in the U. S., developed by Allmanna Svenska Electriska Aktiebolaget, of Sweden. The high frequency induction furnace is cataloged in eleven sizes with nominal charge capacities from 500 lb to 15 tons. As designed by ASEA since 1926, the furnace is essentially a thin walled crucible surrounded by a specially designed coil of water cooled copper tubing. A large furnace will melt a 12-ton heat at 5 tons per hr.

A new "soft" magnetic material composed of 16 pct aluminum and 84 pct iron has been developed by the Navy Ordnance Laboratory. Named 16-Alfenol, it has properties superior to silicon iron and predictions are that it will be especially useful in transformer cores used in high frequency communication instruments. The alloy itself is not new and was used by the Japanese during World War II.

A time-saving scarfing system may speed up stainless steel production. Linde Air Products Co., a division of Union Carbide & Carbon Corp., developed the system being tried out at Atlas Steel Ltd.'s Welland, Ont. plant. The Lin-de-Surfacer, contains a cutting flame into which powdered iron is introduced, building up super-hot flame temperatures. The burners operate on all four sides of the billet or slab, removing surface imperfections. Depth of the cut is regulated by speed of metal passing through machine.

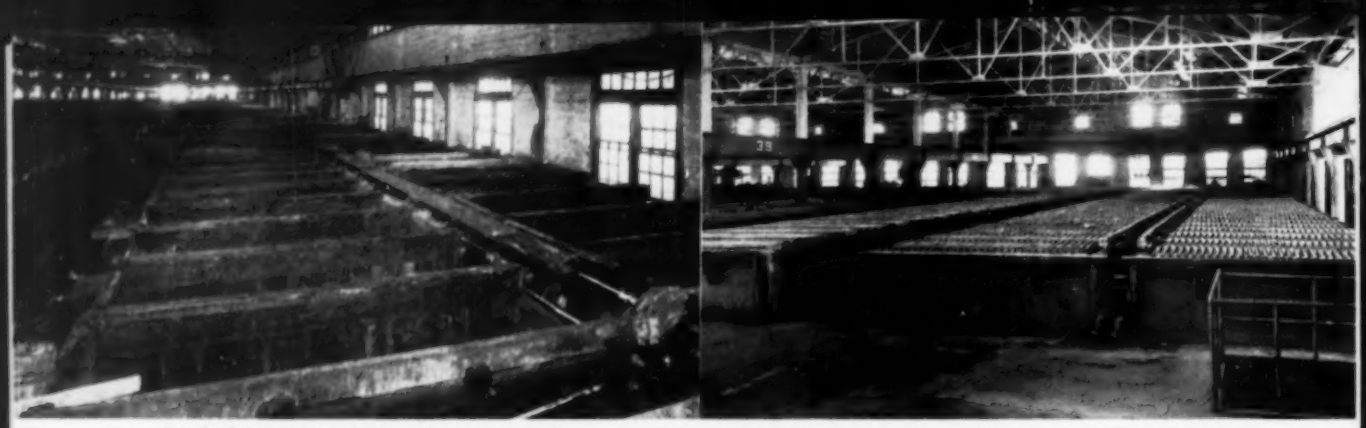

Fig. 1—Rehabilitation of the tank house resulted in a more pleasant place to work, as well as providing new and better equipment. The old wood section showing deterioration of tanks as well as the drabness of the working area can be seen at the left. At the right is a view of the tank house looking west, showing new concrete sections under current. Excellent visibility is achieved aided by the white walls, ceiling and trusses.

ASARCO Rehabilitates Copper Refinery

Perth Amboy Plant Improves Operating Efficiency And Employee Morale

by G. H. Weis, D. A. Busch, H. K. Spaulding, G. B. Paulding

AFTER thoroughly inspecting the physical condition of its plant, Perth Amboy, N.J. organization of the American Smelting & Refining Co. reached the conclusion that the time was at hand to proceed with a complete rehabilitation of the copper refinery from the cellar to the roof. Over the years—especially during the second World War, from 1942 to 1946—the condition of the electrolytic refinery and the auxiliary units deteriorated to an extent that economical maintenance could not be achieved. In addition, work efficiency and employee morale fell to a low ebb. Main operation of the plant, which covers approximately 150 acres, is the refining of lead and copper bullion.

Rehabilitation of the copper refinery, consisting of the electrolytic tank house, slimes dept., copper removal and electrolyte depts., started in 1946 after conclusion of the four months nation-wide strike of the copper industry. The results of the program — increased production and employee efficiency — as well as a preventative maintenance program that will assure efficient operation for many years, attest to the wisdom of the company decision. Views of the tank house before and after rehabilitation are shown in Fig. 1.

The electrolytic tank house, employing the multiple system, is contained in one large room covering an area of approximately four acres. The general arrangement of the tank house before rehabilitation can be seen in Fig. 2. At the start of the rehabilitation there were 48 electrolytic sections of 34 wood tanks lined with 6 pct antimony lead, which had been in service for approximately 25 years. Normally 41 sections, or a total of 1394 tanks, were used for producing commercial cathode copper; four sections, or a total of 136 tanks, for the production of brittle cathode copper for an extrusion process, and three sections, or a total of 102 tanks, for the production of starting sheets. There was one central pump or machinery bay 18 ft wide extending the full length of the tank house. The circulation equipment in the pump bay consisted of four horizontal centrifugal pumps for all the commercial tanks, and one each for the brittle cathode and starting sheet tanks. Each circulation was equipped with its own head and sump tanks. The circulating electrolytes were heated by low pressure steam through horizontal heat exchangers. Adjacent to and at one end of the electrolytic house was located the slimes treating, copper removal and electrolyte purification depts.

The slimes treating plant consisted essentially of two rectangular receiving-settling tanks for the daily receipt of raw electrolytic copper slimes and separation of the electrolyte and wash waters. The leaching equipment comprised three wood stave lead-lined tanks equipped with agitators and air blowpipes. A plate and frame press was used for final filtering of the leached slimes before transfer in four-wheel carts to the cupel dept. for further treatment. The electrolyte separated from the raw slimes, leaching solutions, and wash waters was passed through a series of three wood lead lined tanks to permit further settling of solids and then passed through a plate and frame filter press using paper as a filtering medium for final polishing before return to the electrolytic refinery as makeup solution.

The copper removal unit consisted of six sections of four tanks each for control of the copper content of the tank house electrolyte. Two sections of four tanks each and six single tanks in cascade were used for the removal of copper from foul electrolyte be-

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fore delivery to the purification dept. for the removal of nickel and recovery of contained sulphuric acid. All tanks in this unit were wood, lead lined, and equipped with insoluble lead anodes and copper starting sheets to complete the operating cells. A wood lead lined sump and head tank and horizontal centrifugal pump comprised the major equipment of the circulating system.

The electrolyte purification unit equipment consisted of one vacuum evaporator with internal heat exchanger, four cast iron oil-fired boiling pans, a settling launder for separation of nickel sulphate from the concentrated sulphuric acid, and a suspended type centrifuge for washing and drying the crude nickel sulphate for the market.

To determine the magnitude of the program about to be undertaken, several survey teams were set up, consisting of members of the plant engineering, development, maintenance, construction and operating depts. These survey teams, after a thorough inspection of the various units and areas, reported findings and recommendations to management.

A large portion of the main roof and ventilation monitors were in need of replacement, as was the entire anode receiving aisle.

Inspection of crane runways revealed that the beams had become fatigued by long service and the steadily increased loads being handled.

Deterioration of the wood electrolytic cells had progressed to the extent that two thirds of these tanks would require immediate replacement.

The machinery and pump bay equipment were in need of complete replacement, as were the slimes sumps and pumps. Incorporated in the program for rehabilitation of the machinery and pump bay was the replacement of the heating steam condensate reclaiming tank. It also was found that the cathode wash water system could be incorporated advantageously in this portion of the project.

During the early discussions concerning the extent of the program to be adopted, it was found that the brittle cathode production unit could be relocated economically during rehabilitation, and thus relieve a most undesirable employee morale situation developed because of the poor location.

It was determined that the entire copper removal unit and electrolyte purification unit would require extensive and in some cases complete replacement of tanks, electrolytic cells, pump sumps, drain tables and settling launders.

The slimes receiving and leaching tanks with agitating equipment and other miscellaneous equipment were found to be beyond economical repair.

Hydrostatic pressure beneath the cellar floor had long been a problem in keeping the cellar dry. Solution of this problem, it was found, could be incorporated with work to be done on the slimes sumps.

During the entire formative period of the program the proper protection against acid corrosion of the new and rehabilitated equipment and areas, as well as the older equipment, was of major concern.

Preliminary Planning

Reports and recommendations of the survey groups were studied carefully and findings turned over to an engineering group. Many preliminary drawings and cost estimates were prepared by this group. These preliminary drawings and cost estimates along with the survey team reports enabled management to reach decisions concerning scope of the rehabilitation program, and the process changes that could be reasonably incorporated within the program.

Considering the entire project as one of deferred maintenance, a general policy was formulated that precluded the wholesale adoption of new processes and methods. However, this policy was modified where a new process or method would lend itself to better preventive maintenance.

Where new equipment was required to implement the program, the policy adopted for selection included the best and most effective safety features known for employee protection, ease of inspection for deterioration and wear, economical repair at a later date, and construction materials that would offer the greatest resistance to acid corrosion.

During the period that this project was undertaken, the domestic demand for refined copper production was at an extremely high level. This demand ruled out the shutting down of the copper refinery during the rehabilitation. Therefore, a major objective of planning, both before and during the operation of the project, was to maintain maximum refining capacity without seriously delaying or impeding actual construction work. The refinery is normally rated at 14,000 tons of refined copper per month. Extent of the cooperation between the operating and construction groups is indicated by the fact that the average monthly cathode output approximated 12,750 tons during the rehabilitation.

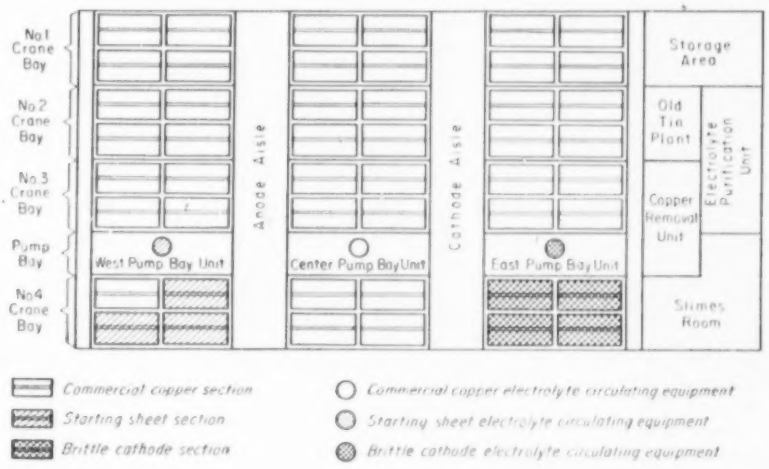

Fig. 2—This outline of the copper refinery building shows the general arrangement before rehabilitation. Each of the 48 electrolytic sections consisted of 34 wood tanks. Deterioration of these cells had progressed to the extent that two thirds of them required immediate replacement.

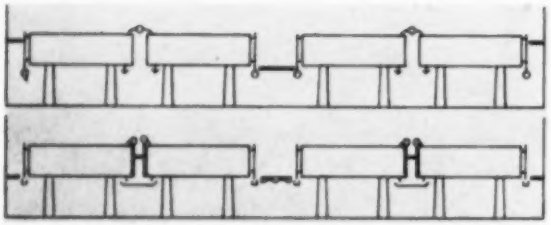

Fig. 3—Top figure shows the old wood section electrolytic cell arrangement, with the separate *V* slimes launders and common feedline. This was replaced by a new concrete section arrangement, bottom figure, with wide single slimes launder, dual feedlines and feedline walkway.

As the scope of the undertaking unveiled, it was realized that an organization separate from the plant maintenance crew would be necessary to effectively handle the job. An engineering group consisting of a mechanical engineer and four draftsmen was set up to record ideas of all concerned. Preliminary drawings were prepared, discussed, revised, and the overall plan finalized.

With a working plan at hand, the engineering group was assigned the dual task of preparing necessary working drawings and procuring all material and equipment. Drawings were prepared and materials procured in the proper sequence. This complete engineering of the job did not delay the start of actual construction. During the project the engineering and procurement progressed with, but ahead of, construction.

In setting up the construction crew, streamlining as much of the work as possible was a primary factor. Much of the work involved in the project was to be repetitive. This was especially true in the tank house proper where a major part of the work was section rebuilding.

With this thought in mind the construction crew was divided into two distant groups. One group was assigned to the fabrication of certain components, required in quantity, which could not be secured in their final form directly from suppliers. Falling into this category were tie rods, feedline and launder supports, cast lead fittings, fabricated lead fittings, feedlines, wood tank bottom boards, concrete walkway slabs, etc. All crafts in this group with the exception of the lead burners were available and they could be drawn from the plant maintenance dept. It was estimated that 30 lead burners would be required. Starting with six qualified burners and an experienced supervisor, an accelerated training course was established to train the balance of the burners required. Trainees were drawn from likely prospects among plant employees. Most of them were burning lead within two months. This was one of the high spots of the program and was proof to management that the leadership and team spirit necessary to accomplish the undertaking were present in the plant. A second construction group was assigned to the actual erection work. A nucleus for this group also was drawn from the plant maintenance dept. At the peak of the rehabilitation program there were approximately 200 men working on the project. A portion of the work was performed by outside contractors. For example, concrete tank components were supplied by an outside contractor on a rigid schedule.

Fig. 4—This view shows the complete state of demolition required before reconstruction could begin on a group of sections.

Although considerable time had been devoted to working out in advance as many of the problems as could be foreseen, it was obvious that some revision was required. Accordingly, initial time schedules were made more or less flexible. Originally it was intended that the rehabilitation program be carried on under the direction of the plant master mechanic. However, because of other duties, it was not practicable. An engineer from the plant engineering office was assigned to supervise construction on a full time basis, reporting to the plant engineer.

At this stage sufficient experience had been gained to make possible the establishment of a fairly rigid schedule of operations. This schedule was of considerable help in expediting materials and distributing the workload among various construction groups.

As the work progressed many valuable suggestions were received from the workers. The most noteworthy of these was an idea submitted by a carpenter group leader that, upon adoption, permitted the installation of the concrete end slabs in the new electrolytic tanks without using the overhead cranes. During the entire construction program the availability of tank house cranes had a serious effect upon work schedules. The overhead cranes were required for refinery operation as well as for the handling and erection of building materials.

Electrolytic Cells

The old wood electrolytic cells to be replaced were designed in two nests of 17 tanks each, constituting a single section. The wood used for the electrolytic cells was long leaf yellow pine lined with 6 pct antimonial lead. The tanks were designed so as to have a common partition wall between tanks and each resting on long leaf yellow pine sills set on cast concrete piers. The electrolytic tanks were equipped with top-to-bottom circulation flow and a single feedline between the two nests to feed the individual tanks via nipples and Para rubber tubing. Fig. 3 compares the old wood section arrangement with the new concrete section.

The return solution collectors were constructed of 6 in. diam pipe with suitable openings equipped with boots to receive the overflow solution from each electrolytic tank. There were two slimes launders of *V* type construction, one each under a nest of 17 tanks to receive the slimes during the unloading operation. The crane aisle walkway was of wood plank construction and at the same elevation as the anode and cathode cross-aisles. However, the back aisles were of wood slat construction and were at an elevation approximately 12 in. below the tank caps. It can be seen from this description that some

changes were required to prevent accelerated deterioration of the cellar by providing double feedlines for each section with a common slimes launder directly below the feedlines. In this manner spillage caused by hose breakage and line leakage would fall into the slimes launder rather than to the cellar floor as had heretofore been the case.

Before rebuilding the sections all acid corrosion of the cellar floor was repaired by cutting away the damaged areas and replacing with concrete. In many cases acid corrosion had proceeded to the point where the tank piers were left without suitable footings. These were removed and rebuilt before placing the new concrete tanks. The complete state of demolition required before reconstruction could begin is shown in Fig. 4.

Design of the new concrete electrolytic tanks was such as to make precasting possible and then final assembly in place. Adjacent tanks utilized a common partition and cap between cells as can be seen in Fig. 5. The back and front ends of the tanks were made of three sections of precast concrete slabs. Long leaf yellow pine planks were used for bottoms. The tank linings were of 6 pct antimonial lead. Each electrolytic cell was equipped to provide either top-to-bottom or bottom-to-top circulation flow. Vitrified ceramic tiles were substituted for the pine capboards previously used. In placing the nests of tanks making up the complete section, space was provided between the nests for a removable walkway, which provided easy access for operations, inspection and maintenance work. The back aisles were lowered to the same elevation as the main aisles, which also provided easy access for inspection and maintenance work. To provide for routine and efficient cleaning of return solution collectors, open top phenolic resin launders were installed. Company experience at another electrolytic refinery was utilized in making this selection. Performance to date has been entirely satisfactory with practically no breakage during operation and little or no current leakage because of dielectric properties. The design of the slimes collecting launder provided a flat bottom and sloping sides with inverted lips at the top of the sides to prevent splashing of the refinery slimes during tank cleaning operations.

The seepage of subsurface water into the tank house cellar had constituted a major operating problem in the past because of hydrostatic head and cracked cellar flooring. It was deemed impractical to attempt waterproofing four acres of cellar floor. Therefore, six subsurface sumps were located strategically within the cellar area. It was also desirable to provide greater head room at the slimes collecting sumps, providing more slope to the main slimes collecting launders from the various sections to these sumps. The final design combined the construction of the slimes sumps and pump pits adjacent to the subsurface drain pits. Automatic pumps were provided to drain the subsurface pits. The resultant solution was piped to an iron cementation tank for the recovery of contained copper. To prevent the loss of refinery slimes, emergency overflow sumps were provided adjacent to the operating slimes sumps. These sumps were suitably equipped with electrical signaling devices to provide ample warning in the event of power failure or pump line blockage. Operating and maintenance experience to date has been entirely satisfactory with the design and construction features incorporated in these units. In replacing the refinery slimes pumps, horizontal pumps were adopted. The previous pumps in this service were vertical boot pumps that required heavy maintenance expenditures.

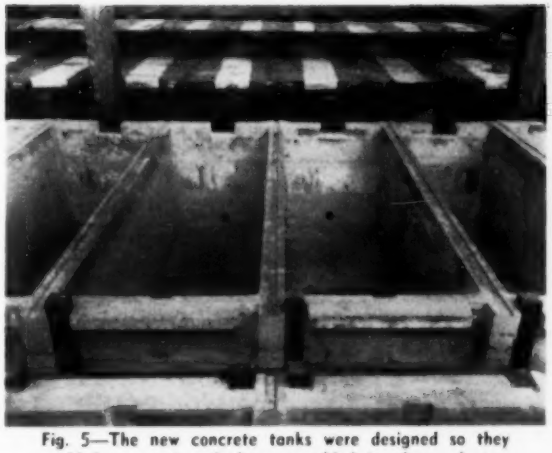

Fig. 5—The new concrete tanks were designed so they could be precast and then assembled in place. A new section can be seen under construction in the foreground, with the old wood section in the background. Adjacent tanks have a common partition and cap between cells.

Pump Bays

The pump bay is the area in which the pumps and associated equipment necessary for the circulation of electrolyte in the copper refinery are located. The anode and cathode aisles that run transversely through the tank house divide the pump bay into three units.

Prior to rebuilding, the east unit housed the circulating system for the brittle copper producing sections that were located in the southeast corner of the tank house building. This circulating system consisted of a rectangular lead-lined wood collecting tank for return solution from the sections, a centrifugal pump for elevating this solution to a rectangular lead lined wood head tank, and a horizontal tubular heat exchanger for heating the solution being fed to the sections from the head tank. In addition to this equipment, the east unit contained one of the six slimes sumps complete with vertical boot pump as well as a cylindrical lead lined wood condensate storage tank and a rectangular lead lined wood solution collector for the return solution from the 12 commercial sections at the east end of the tank house.

The west unit of the pump bay contained the circulating system for the three sections located in the southwest corner of the tank house used in the production of starting sheets. This consisted of essentially the same equipment as used in the brittle copper circulating system. This unit also contained one of the six slimes sumps as well as a collector for the return solution from the 13 commercial sections in the west end of the tank house. In addition to this equipment, this unit contained the collecting tank and pump for transferring Pyne-Green solution to the copper removal system.

Before rebuilding, the middle unit contained the circulating equipment for all of the commercial sections in the tank house with the exception of the two collecting tanks located in the east and west units. This equipment consisted of one large rectangular lead lined wood collecting tank for the return solution from the 16 commercial sections and was interconnected with the other two commercial

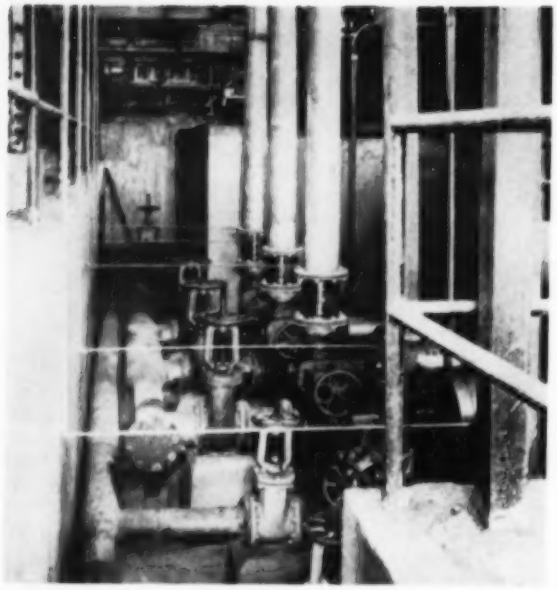

Fig. 6—This general view shows the pumps and concrete sumps in the pump bay unit.

electrolyte collectors by means of equalizing lines. There were four centrifugal pumps for elevating all commercial solution to a large cylindrical lead lined head tank from which were fed the 41 commercial sections in the house through 17 horizontal tubular heaters. This unit also contained a slimes sump and pumping system and in addition, the cathode wash water handling system.

Plans for the rebuilding of the pump bay called for the duplication of all equipment used in each of the circulating systems so uninterrupted solution flow could be maintained during repair periods. Plans also called for dividing the one large commercial circulation, located in the middle unit of the pump bay, into three separate circulations—one located in each of the other pump bay units.

This meant that ultimately there would be five circulating systems in the tank house:

1—Circulation for the 12 commercial copper sections in the east area

2—Circulation for the four relocated sections for the production of brittle copper, also in the east area

3—Circulation in the middle area for the 16 commercial sections

4—Circulation for the three sections on starting sheet production in the west area

5—Circulation for the 13 commercial sections, also in the west area

It was decided that the east section would be rebuilt first, followed by the west section and then the middle section. Fig. 6 shows pumps and concrete sumps in the pump bay unit, and electrolyte heaters in one pump bay unit can be seen in Fig. 7.

During construction of the east area, the commercial sections were fed in the usual manner from the head tank in the middle pump bay unit. All but a small portion of the east return solution collecting tank was removed to clear the area for new construction. The commercial sections still in operation in the east end of the refinery were then connected to the remaining portion of the collecting tank through temporary lines. Since, as mentioned previously, this tank was connected to the central solution collector and main circulating pumps through an equalizer line, circulation of commercial electrolyte was maintained at all times. The area thus cleared was sufficient to accommodate a part of the new concrete pump system planned for the east bay.

After underpinning all building columns surrounding this area, the dual sump tanks for both commercial and brittle copper electrolytes, the large emergency sump, the dual slimes sumps, the subsurface water sump and a large enough section of the pump pit to permit the placement of one circulating pump for each electrolyte, were installed.

The balance of the new equipment for the brittle copper circulation then was installed and put into service. This equipment consisted of two new cylindrical lead lined wood head tanks and two new horizontal heat exchangers complete with all necessary piping. All cylindrical tanks installed during the rehabilitation were of open-stave construction for ease of maintenance.

With this unit in service dismantling of the old brittle copper circulation equipment was accomplished to make room for the rest of the new equipment for the commercial copper circulation, plus two new cylindrical lead lined wood storage tanks, one for each electrolyte to be stored. This circulating equipment was essentially the same as that for brittle copper except that the head tanks were of larger capacity and four heaters were required instead of two.

With this work completed the commercial unit was put into service and the remaining portion of the old solution collector was removed. This allowed further underpinning of the building columns and the extension of the pump pit to accommodate spare pumping equipment. The reconstruction of the west pump bay was accomplished in much the same manner.

Upon completion of the circulating systems for the east and west units of the pump bay, rebuilding of the center circulation system serving 16 commercial sections was started. Since this unit of the pump bay contained equipment for only one circulation, a large concrete condensate collecting tank and

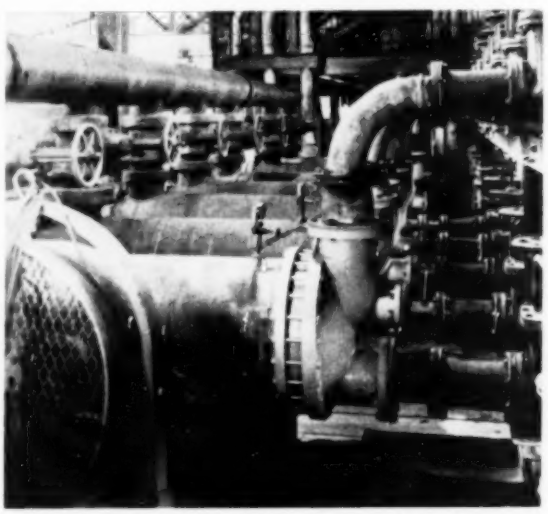

Fig. 7—Electrolyte heaters in one pump bay unit are shown in this photograph.

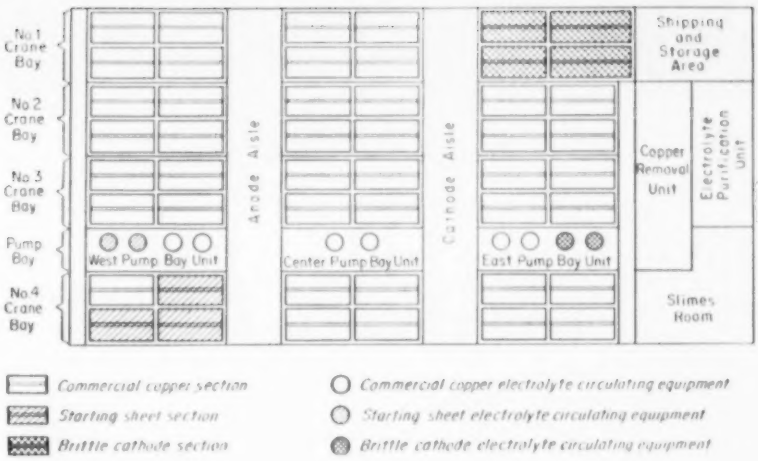

Fig. 8—This outline shows the copper refinery after rearrangement of pump bay units and relocation of the brittle copper unit. This relocation was accomplished in 2 weeks, and resulted in increased productivity of the unit.

a concrete collecting tank and pump for Pyne-Green solution handling were incorporated with the new concrete sump system and pump pit. In addition to this equipment, a new cathode wash water system was installed under the cathode aisle. As before, extensive underpinning of building columns adjacent to the new concrete sumps was necessary.

In this area new construction proceeded slower than heretofore. Much temporary work was required before the final arrangement was attained, as there was a greater quantity of equipment present at the start and many moves were necessary to keep the refinery in continuous operation.

The main anode aisle floor, originally constructed of double-planked yellow pine flooring on timbers and cast iron Lally columns with occasional auxiliary supporting steel members, had passed the stage where further patching was possible. The entire floor, some 9000 sq ft, was replaced with reinforced concrete cast in place around new structural steel members, supported by new steel columns on new concrete footings. The design was engineered for heavy loading, as the live load on the narrow gage trackage exceeds 600 lbs per sq ft. The tracks were placed with the top of the rails flush with the floor and trolley-type switches were used to eliminate tripping hazards. Construction was accomplished while operations continued by staggering the unloading so that anodes for sections in the crane bay of whichever floor section was being rebuilt, could be brought in through the cathode aisle. Hi-early concrete mix was used to speed up the curing time. Each section of floor required about one month to dismantle and reconstruct.

As mentioned before, large sections of the main roof were in need of complete replacement. As experience previously indicated, wood decking with 20-year built-up roofing proved satisfactory. However, the older roof decking was supported on wood purlins. In the rebuilding of the main roof sections, the only substitution made was steel purlins for the wood purlins. Several small sections were replaced with a calcium silicate roof decking for experimental and test purposes. Experience to date indicates that calcium silicate will be an excellent substitution for wood decking and will provide the added feature of being fireproof. In replacing the ventilation monitors, corrugated glass was substituted for the wood decking and built-up roofing to provide natural lighting of the tank house interior. An added feature realized by this substitution has been the reduction of maintenance costs of the monitors.

It was necessary to replace all crane runways in the tank house. Continuous service over the years and increased crane loads as the refinery expanded caused settlement of the building columns as well as fatiguing of the crane beams. New crane runways were designed for future cranes capable of handling maximum anticipated loads of 12½ tons.

As there was as much as 5 in. difference in elevation between high and low points over the length of the runways, it was necessary to take levels on all of the beam brackets before having the beams fabricated.

All of the dismantling of the old runways and the erection of the new was done at night so as not to interfere with crane service to the operating copper sections. Any work during this phase of the construction that was started after suspension of refinery operations for the day had to be completed in time for the next day's operation.

Two crews were used, one on each side of the bay in which the beams were being replaced. Although few of the beam connections were the same because of the varying elevation of the beam brackets, the preliminary survey, accurate fabrication of all parts and a system of identification made it possible to do the work within the time required.

About 10 pct of the refinery output is produced as brittle cathodes on a toll basis and provides the refined copper for coalescent production of another company. The expansion of production requirements for brittle copper over the years resulted in an acute materials handling problem as both ingress and egress to the unit in its original location was via one crane. This resulted in production delays and reduced monthly average output. In planning to rehabilitate the electrolytic sections in use for this work and maintain normal production, it was necessary to move the auxiliary equipment required in this operation. Therefore, an area was selected in which two or three cranes were available and so located that the anodes could enter at one end and the refined brittle cathodes be shipped from the other end. Experience gained in the operation of this unit during the expansion years was utilized in laying out the operating equipment to provide well-ventilated and well-lighted work areas for the employees. Every known safety feature was incorporated in the design and layout to provide the ut-

most safety to the workers. The relocation of this unit was accomplished within two weeks and with a loss of less than 8 pct of the normal production during the moving period. The wisdom of the decision to move this unit during the rehabilitation process has been borne out by the increased productivity of the unit and the improved employee morale experienced to date. The general arrangement of the tank house after rehabilitation is shown in Fig. 8.

Prior to the rehabilitation of the refinery, attempts were made to protect the wood ceiling from acid fumes with oil-base paints. Examination of coatings applied within the past 10 years indicated that oil-base paints were not entirely satisfactory. Earlier in the program various coatings were applied as test panels for observation. As it was desirable to provide well-lighted working areas within the tank house, white was selected as the color to be used. In the short space of time provided for paint testing, the conclusion was reached that an oil-base silicate pigmented paint was the best protective coating available at the time. The major portion of the ceiling had been painted when it was learned that the alkyd-base paints would be superior in lasting qualities than oil-base paints. Midway during the painting program the alkyd-base paint was adopted.

For many years asphaltic paints were used for the protection of operating wood and steel equipment. As this could only be obtained in black, the appearance of the electrolytic tank room was very drab. With the substitution of concrete for wood in the construction of electrolytic tanks and sumps, it became possible to adopt chlorinated rubber-base paints. As these were obtainable in various colors, a color scheme was adopted and standardized. All concrete tanks were painted with a light green protective coating, steel building structures were painted dark gray, operating structures and equipment light gray, and cranes and moving equipment were painted orange. The protection provided for the equipment against direct acid contact and fume corrosion has amply justified the use of chlorinated rubber-base paint as a protective coating. A further dividend has been a marked improvement in refinery employee morale.

The slimes room, consisting essentially of receiving, settling and filtering equipment to process and return the solution and wash waters inherent to the process to the tank house, as well as leaching and filtering equipment to process the electrolytic slimes preparatory to smelting, was rehabilitated without affecting normal refinery operations. This was accomplished by maneuvers such as constructing a launder through one old receiving tank which had to be dismantled. Slimes and solution were passed through this launder while the old tank was torn down and two new tanks built in its place. One key pump had to be moved and reconnected six times before being placed on its permanent foundation.

Four new conical bottom, overflow rim, 750 cu ft capacity receiving tanks were constructed. Three existing tanks were revamped as settlers and one new settling tank was constructed. The flow through the system was rerouted for better process continuity.

A 1000 cu ft leaching tank was brought in from another part of the plant and installed in a temporary position to take the leaching load while permanent installations were being made of two additional 1000 cu ft tanks. The first temporary tank eventually was moved to a permanent location. These steel tanks were lead lined and then inner lined with acidproof brick laid in a resinous bond. The new steel leaching tanks were installed at a considerably higher elevation than the old equipment, thereby facilitating discharge by gravity flow out of the tanks when leaching is complete, as it is important to be able to make a clean cutoff between charges for effective process control.

The copper removal unit, made up of six sections of four tanks each for copper control service, two additional high-circulation sections and a six-tank row of cascade series-flow cells for preparation of solution for the purification unit, was enlarged and completely rebuilt. The cascade cells were moved to the next bay, reconditioned and eight more cells added, making a total of 14 arranged in two parallel rows of seven cells, as shown in Fig. 9.

The cellar floor was resurfaced, tank piers were reinforced, tanks and linings replaced, pumping facilities expanded, sump and head tanks rebuilt.

Fig. 9—The six-tank row of cascade series-flow cells for preparation of solution for the purification unit was moved, reconditioned and enlarged to two parallel rows of seven cells. The cells were hooded with individual stainless steel hoods to protect personnel from the hazardous and annoying gases.

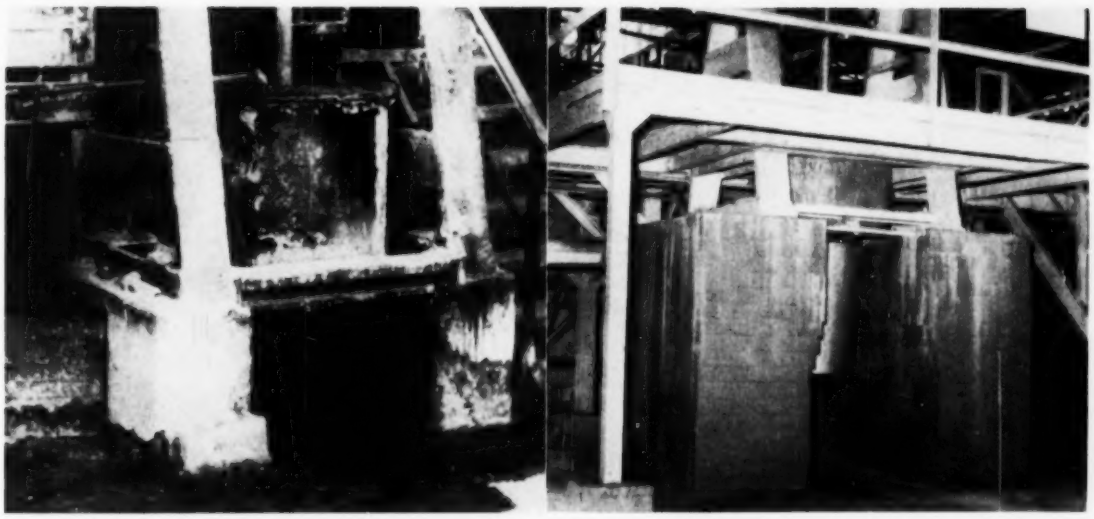

Fig. 10—Typical conditions before rehabilitation are shown in the view at the left of the original centrifuge installation in the purification plant. This gave way to the new centrifuge installation shown at the right. The improvement in working conditions can easily be noted.

The unloading of cathodes from the copper control cells was simplified by developing a simpler element lineup in the sections and by employing a wash tank and stacking rig within the unit instead of depending on the main tank house cathode loading machines. This unit uses lead anodes. The casting of these anodes was greatly simplified by converting from a closed vertical book mold to an open horizontal mold.

As the evolution of gases from the cascade tanks is both hazardous and annoying to personnel, these cells were completely hooded with individual stainless steel covers connected to a power-driven exhaust system. Several types of alarm systems give ample warning in case of loss of draft.

The Perth Amboy plant is a custom refiner and it is the practice to accept for refining a large variety of blister copper, anode copper as well as scrap copper. It is therefore essential to have electrolyte purification equipment designed to handle a sustained large volume of electrolyte requiring treatment for the removal of sizeable quantities of impurities. The existing purification plant had been inadequate for the purpose and had deteriorated to the extent that it had to be completely rebuilt. In addition, one whole part of the old process was eliminated as being impractical to operate on a plant scale. It was decided that increased capacity of other sections of the process would amply take care of the impurity removal previously expected of that part of the process which was eliminated.

For the first stage of the process new vacuum evaporators were designed. One of the most important features was the external vertical heat exchangers, so designed as to be readily accessible for regular cleaning. An extra heat exchanger was provided and all exchangers were connected to the evaporators and pumps so that one heater at a time could be cleaned without affecting the operation of any evaporator.

The second stage of the evaporation is carried out in oil-fired cast iron pans. The design of these units was based on the layout in another of the company's refineries, and has proven quite successful.

Sufficient settling capacity for separating the concentrated acid from the nickel salts was built into the system. An additional centrifuge was added to the unit, and the original centrifuge was modernized with a stainless steel shell. Fig. 10 shows the original centrifuge and the new centrifuge installation.

A key factor in the operation of multiple units, some of which are continuous, others batch, is sufficient storage tank capacity between process units. The continuous flow from the two rows of cascade tanks is collected in separate sumps, which are pumped individually and automatically to one of three 1500-cu ft storage tanks. Another of the three storage tanks serves as the feed tank for the vacuum evaporators. This separation facilitates accurate solution volume measurement. The vacuum evaporators are fed automatically and continuously by their own vacuum. They are discharged continuously by pumps working against their vacuum. The level in evaporators is automatically maintained by an overflow weir. The discharge is checked for specific gravity, and the operator adjusts this by opening or closing the feed valve in small increments. Storage tanks following the vacuum evaporators provide the necessary surge capacity for the next step in the process, batch evaporation in the cast iron pans. Settling, centrifuging and acid pumping facilities are all protected in the same manner with required storage capacities at strategic points in the process.

The importance of good design and proper plant maintenance is stressed. The absence of one or both of these important elements eventually requires the herculean task described. Fifteen months and a very large expenditure of money were required to bring this plant back to good operating efficiency. The results obtained are indicated by the fact that the man-hours consumed per ton of copper refined have been brought back to prewar standards or better. Furthermore, indications are that the exacting preventive maintenance procedures put into effect will insure an efficient operating unit for many years to come.

Air Force Heavy Press Program Moves From Planning Board Into Fabrication

by Alvin S. Cohan

Editor, JOURNAL OF METALS

THE U. S. Air Force heavy press program, late in starting, is finally well under way. Founded on the need for large forgings, resulting in stronger and lighter aluminum components for aircraft, the program calls for forging presses larger than have ever been constructed. Largest presses in operation in this country are the 18,000-ton press of Wyman-Gordon at North Grafton, Mass., and the 15,000-ton press at Alcoa's Cleveland works. Russia has the largest known press abroad, appropriating the 35,000-ton unit from Germany. Eight aluminum forging presses, ranging from 25,000 to 50,000-tons capacity were ordered by the Air Force.

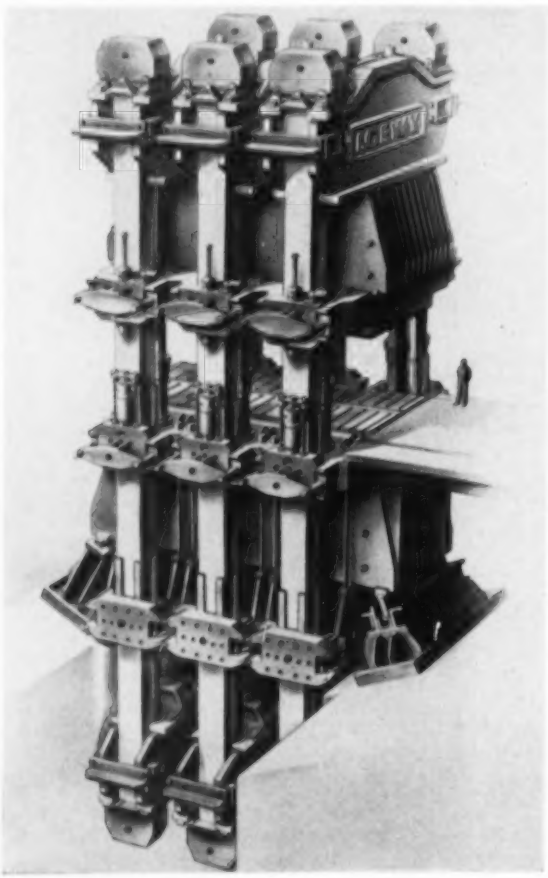

Fig. 1—This perspective drawing was made from the blueprint of the Loewy 50,000-ton hydraulic closed die forging press. The press will weigh 10,750 tons, with an overall height of 114 ft.

Two 50,000-ton and four 35,000-ton presses are on order. One of each was designed, and is being engineered and built by Loewy Construction Co., a subsidiary of Hydropress, Inc. They will be installed and operated at the North Grafton, Mass. plant of Wyman-Gordon. Design of the presses was no simple problem. Design data for so large a project was, of course, unavailable. Where complex design calculations could not be made, experimental methods were resorted to—including two and three dimensional photoelastic studies, and scale model testing. Experimental stress analysis work was performed by Loewy Construction Co. utilizing the laboratory facilities of Purdue University, Alabama Polytechnic Institute, New York University, and University of Illinois.

The 50,000-ton forging presses will be the world's largest. A perspective drawing of the Loewy press, *Major* is shown in Fig. 1, and the 35,000-ton press, *Minor*, in Fig. 2. Design data for each press is given in Table I. The larger press will have 9 main hydraulic cylinders and six press columns. The smaller press will have eight hydraulic cylinders and four press columns. All structure under the lower platen of the press will be submerged below the floor level.

The presses are designed so that all structural members are in almost pure compression. Presses are of the pull down type—forces are transmitted to the lower platen, and through the main tie rods to the upper entablature, with the entire entablature moving down with the upper platen.

Automatic structure safety devices are being built into the presses. In addition to a hydraulic force compensating system for compound eccentricity, a strain gage system is incorporated. Strain gages will be permanently mounted on the main tie rods, and

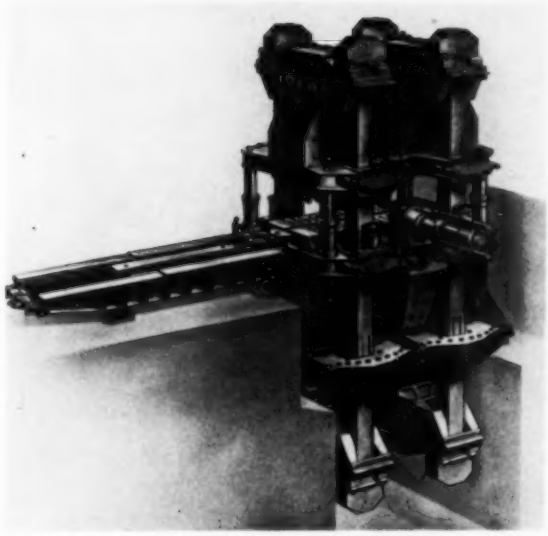

Fig. 2—The Loewy 35,000-ton die forging press is a smaller version of the 50,000-ton press. It will weigh 8550 tons and have an overall height of 107 ft.

Table I. Specifications for Loewy Die Forging Presses

	50,000 Ton Major	35,000 Ton Minor
Capacity, Tons		
With Intensifier	50,000	35,000
	36,000	18,000
	18,750	
Without Intensifier	36,000	24,000
	24,000	12,000
	12,000	
Equipment		
Main Hydraulic Cylinders	9	8
Pullback and Balancing Cylinders	6	4
Eccentric Load Compensating Cylinders	6	4
Press Columns	6	4
Pressure Intensifier, 3:2 Ratio	1	1
Side Cylinders, 3000 Ton	—	2
Operating Data		
Die Bed Area	12x32½ ft.	12 x 30 ft
Daylight, Max	13 ft 9 in.	11 ft 9 in.
Daylight, Min	7 ft 9 in.	5 ft 9 in.
Stroke	6 ft	6 ft
Fast Advance Speed	480 ipm	480 ipm
Return Speed	480 ipm	480 ipm
Sliding Table		
Size	die bed area	die bed area
Stroke	32½ ft	30½ ft
Overall Weight and Height		
Total Weight	10,750 tons	8550 tons
Height, Above Floor	48 ft	45 ft
Below Floor	66 ft	62 ft
Length Tie Rods	108 ft	101 ft

the hydraulic system will be unloaded automatically when preselected strain values are exceeded.

In designing presses of this size, many factors had to be considered. Conventional round forged columns could not be used. Their size was above practical production capacities. Not only would it be impossible to cast ingots of required size, but they could not be forged, machined, or transported. A rather unique solution to the problem was arrived at using columns built up of three rectangular laminations. Each lamination for the 50,000 press is 13½ in. thick and 108 ft 2½ in. long, as illustrated in Fig. 3. This will permit transportation of the members for assembly at the site.

Even this design posed fabrication problems at Bethlehem Steel Co. Each lamination required casting 275 tons of steel in a mold 18 ft high and 9 ft diam. Two open hearth heats, Fig. 4, were used for each ingot, and 18 ingots were required for the huge press. Forging the ingot at 2300°F in a 7500-ton press to form each 145-ton lamination, the largest steel forging ever produced, required a series of five heating and forging cycles demonstrated by Figs. 5 and 6. Special handling equipment and two cranes were required to carry the forging from heating furnace to forge. When the forging reached a length greater than the 37-ft heating furnace, only one end at a time was heated to forging temperature and worked. Special supporting devices prevented the forging from sagging under its own weight.

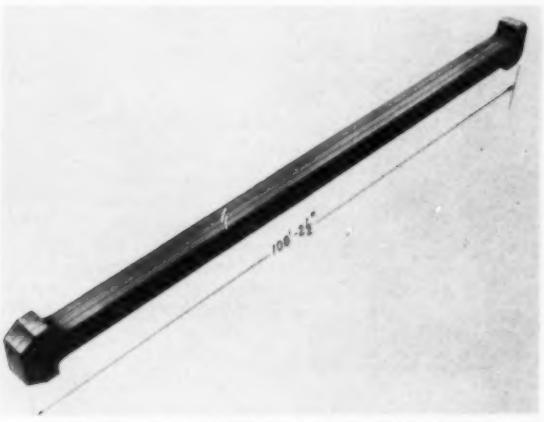

Fig. 3—The scale model shows the construction of one of the six columns for the 50,000-ton press. Each of the three laminations is 13½ in. thick.

Fig. 4—This 275-ton ingot, shown after stripping from the mold, will wind up as one of the three 145-ton components of a single column for the huge press.

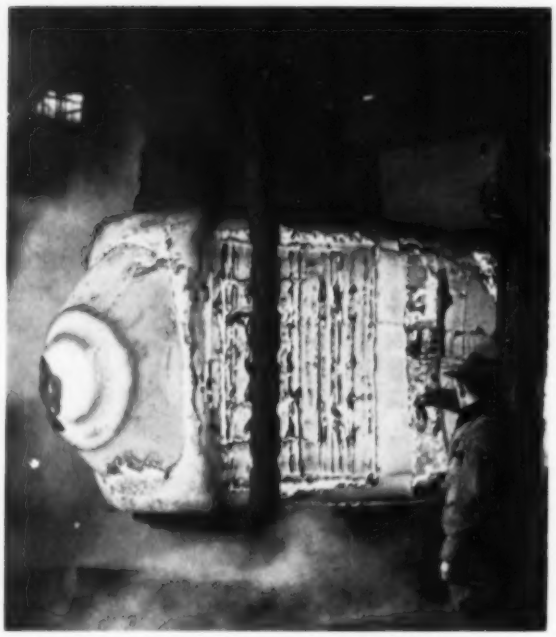

Fig. 5—In this second forging heat in the 7500-ton press at Bethlehem Steel Co., the once round ingot has been forged into rectangular shape and is being elongated.

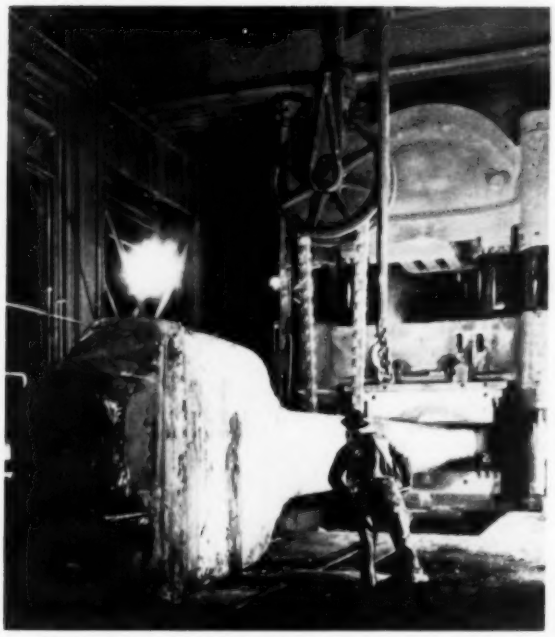

Fig. 6—By the fourth forging heat, the column section has grown to 70 ft in length, and the main body width is being reduced to 44 in. Supporting devices prevent sagging.

When finished, the forged lamination was given a one week heat treatment in a specially constructed 120-ft furnace. Three railroad cars were used to carry the forging to another section of the Bethlehem plant for final straightening in a 6500-ton press, illustrated by Fig. 7. This straightening press had not been used since World War II. After straightening, the forging was annealed in a special furnace over 120-ft long to relieve any stresses set up in the operation.

The press will be installed sometime next year at North Grafton, Mass. where a 100-ft deep foundation, with 13-ft thick concrete supporting walls have been constructed. Plans are being made for the erection of the large and heavy press components. Every conceivable scientific method will be employed for assembling and aligning the presses, and special optical measuring equipment will be used.

Economic production of sound forging for aircraft will not be dependent on the press alone. Forging dies, heating furnaces, and material handling equipment will play an important part. Dies must successfully withstand the largest loads ever produced, and material must be fed to keep the forging giant busy. For each complete cycle, the press advance, power, and return strokes account for only 15 pct of the time required.

Acknowledgment is made for cooperation in preparing this article to the Loewy Construction Co. Inc. of New York.

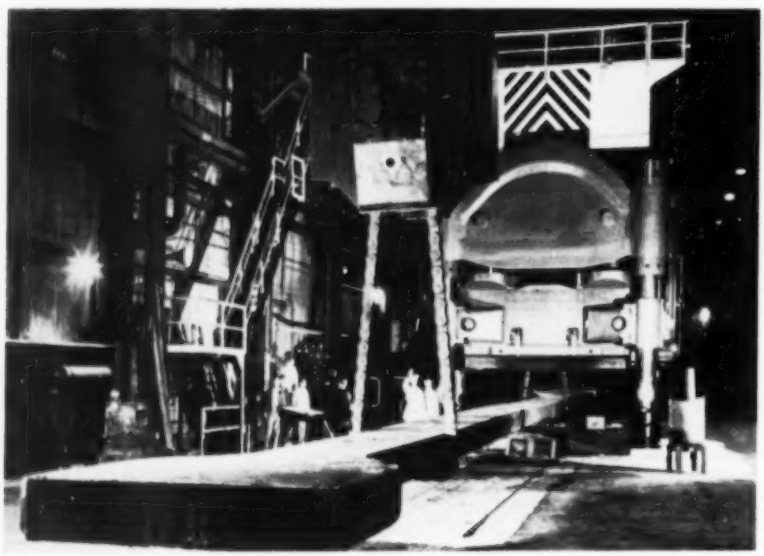

Fig. 7—The completely forged piece, now 110 ft long, is given final straightening in a 6500-ton press. Special furnaces, more than 120 ft long were constructed for heat treating laminations.

Atlantic Steel Using Cold Metal Charge Finds Costs in Favor of Electric Furnace

by J. E. Wilbanks

MELTING facilities at Atlantic Steel Co. consist of an open hearth shop of three furnaces tapping 80-ton heats, and an electric melt shop with one 18-ft JT Lectromelt furnace tapping 75 ton heats.

The electric melt shop was completed and went into operation in May 1952. Since then approximately 1200 heats and 75,000 net tons have been made. There were many problems in getting the electric shop started with open hearth furnacemen who had never seen anything more than a picture of an electric furnace.

The open hearth furnaces operate on an all cold metal charge of approximately 75 pct scrap and 25 pct pig iron. Coke is charged to make up for iron deficiency. The scrap charge is approximately 50 pct purchased scrap.

The type of purchased scrap has not changed since the electric shop went into operation. The bulk of purchased scrap is very light No. 2 country mix and No. 2 bundles. The electric and the open hearth are charged from the same cut of scrap and no preference is given to either shop. The charging of bundles has been discontinued in the electric because of excessive fumes. The fumes, high in zinc oxide, condense and collect on the electrode cooler rings and holders, causing trouble. As for the mill scrap, the electric has not gotten its pro-rata share. Because of light scrap, it is necessary to back charge three to four buckets to get a 75-ton heat from the electric.

During the past six months, the total charge for the two shops compares as shown in Table I. The burned lime charge in the electric was 45 pct less than in the open hearth. Standard practice of charging coke in the open hearth is reversed in the electric by charging roll scale. Enough experience has been gained to know that the performance in tons per hour is one thing with this type scrap, and that it would be much different with heavy mill or electric furnace scrap charge.

Further comparison for the six months period disclosed net metal in the electric was $5 per ton under the open hearth. The electric showed 60¢ per ton lower labor cost. Mold and stool cost was practically the same. This was attributed mostly to tapping by an immersion pyrometer. Otherwise the tendency is to tap hotter heats out of the electric than the open hearth.

Considering fuel, the open hearth showed a $3.85 per ton lower cost. A combination of fuel oil and natural gas is used in the open hearth. Kwh per ton in the electric was 507. Electrode consumption was 9.2 lb per ton, and power rate is 9 mills per kwh.

The open hearth had a $4 per ton lower conversion cost than the electric. The electric showed 94 pct yield against 89 pct for the open hearth. Total ingot cost for the last six months period was in favor of the electric by 25¢ per ton.

Table I. Comparison of Charging Practice of Open Hearth and Electric Furnace

Charge	Open Hearth	Electric
Purchased Scrap, Pct	47	75
Mill Scrap, Pct	22	17
Reclaimed Scrap, Pct	6	8
Pig Iron, Pct	25	—

In looking back over the short time that the electric melt has been in operation, it was encouraging to find that in the last month the cost of ingots was reduced by $18 per ton under the first month. This can only be attributed to experience. It should be noted that both shops have been operating on the same heat schedules and no favoritism has been shown the electric in this respect.

In discussing the economics of an electric vs the open hearth, it is necessary to take a look at repair costs and furnace availability. The electric furnace is now on its sixth lining and sixteenth roof. The furnace is lined with metal encased basic brick and the roof is silica. Average life is 200 heats per furnace lining and 75 heats per roof. No hot patching is done other than routine use of a BRI gun between heats. The roofing is now being redesigned to a water cooled skew back. This, it is hoped, will considerably increase roof life.

A comparison of furnace availability between the open hearth shop and the electric furnace shop during the last seven months showed that the electric furnace operated 94.5 pct of all available hours, while the three open hearth furnaces operated 86.5 pct of total available hours.

Comparison of repair cost, per ton of steel produced, shows that in the same period the open hearths produced 100,603 net tons at an average repair cost of $3.35 per net ton. The electric furnace produced 65,438 net tons at an average repair cost of $2.65 per net ton.

Relining the electric furnace shell from power off to power on averages 12 hr. Most of this time is spent tearing out the old lining and cleaning out the furnace. It takes eight brickmasons about 3 hr to reline the furnace shell, including a new tap hole. Total cost averages $5250. The spare roof ring is lined off the furnace. This requires six brickmasons about 8 hr with a total cost of $1650. To change a furnace roof requires an average of 2½ hr.

A complete repair, excepting a new bottom, will cost about $7000 and can be done in less than 16 hr. Atlantic Steel Co.'s experience indicates that costs are in favor of the electric furnace when making the same grades of steel in an all cold metal practice.

J. E. WILBANKS is Superintendent of Melting, Atlantic Steel Co., Atlanta, Ga. This paper was presented at the AIME National Open Hearth Steel Conference, Buffalo, N. Y., Apr. 21, 1953.

Sinter Sampling Technique Developed By U. S. Steel Corp.

by R. L. Stephenson and D. J. Carney

IN the last few years additional sintering facilities have been installed in both the Pittsburgh and the Chicago districts of the U. S. Steel Corp. Since the construction of these sintering plants made possible the use of higher percentages of flue-dust sinter in blast furnace burdens, it became important to study means of controlling the quality of sinter to obtain optimum results in the blast furnace.

For controlling an operating process, it is necessary first to establish standards by which the quality of the product can be judged. For sinter, it appeared that an important property was its strength or its resistance to degradation during transportation and charging into the furnace. Consequently work was undertaken to establish a standard for sinter strength that could be used both for controlling sintering-plant operations and for correlating sinter quality with blast furnace performance.

The first problem in setting up a standard was that of procuring a sample that would be representative of the sinter made under any particular set of conditions at the sintering plant. Since the U. S. Steel Corp. sintering plants discharge the finished sinter either into a large pit or onto a rotary cooler, the sinter becomes inseparably mixed with material sintered 2 hr before or 2 hr afterwards. For this reason the exact identity of the sinter is lost.

Sampling

The first attempt to procure a sample that would be representative of a specific sinter mix and of specific operating conditions was made by stopping the Dwight Lloyd sintering machine and removing an entire pallet full of sinter. This method, however, proved very difficult to perform and interfered considerably with the operation of the plant. A sampling method was devised by technologists at South works enabling them to secure, without interrupting the sintering operation, a sample of about 1 cu ft of sinter, representative of sinter for the full depth of the sintering bed.

The South works method involves the use of a steel-frame-work basket. The basket has been used both with and without crossbars along the bottom. As long as the crossbars are in the same direction as the grate bars on the sintering machine they do not interfere with the sintering process. The basket is set on an empty grate of the Dwight Lloyd sintering machine before it passes under the swinging feed spout, as shown in Fig. 1. When the basket is removed after it has travelled the length of the sintering machine, it contains the sample. Just before the basket is removed, the sinter is scored and chipped to facilitate removal of the sample from the sinter bed. A view of the basket after its removal is shown

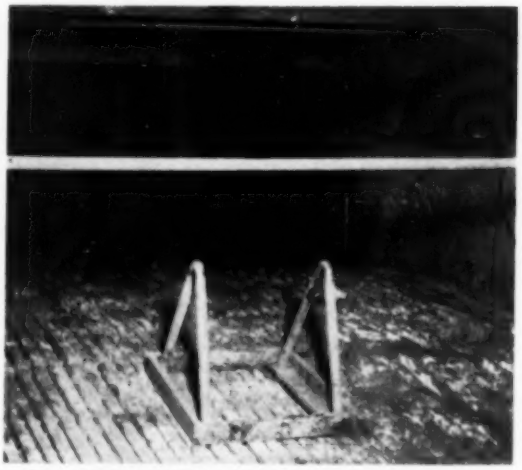

Fig. 1—The sampling basket is set on an empty grate of the sintering machine before it passes under the swinging feed spout.

Fig. 2—After the basket has travelled the length of the sintering machine, it is removed, containing sinter sample.

R. L. STEPHENSON and D. J. CARNEY are, respectively, Research Associate, United States Steel Corp., Pittsburgh, and Metallurgical Engineer, United States Steel Corp., Chicago.

This is an abstract of the article that appeared in MINING ENGINEERING, March 1953.

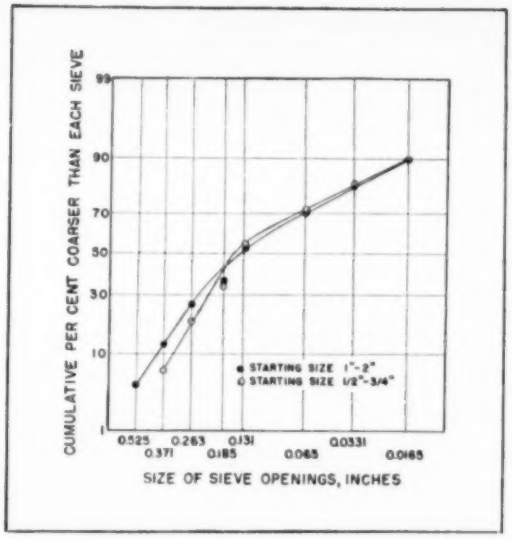

Fig. 3—The effect of starting size on tumbler-test results is shown. It can be seen that the starting size had little effect on the median size of the sinter after tumbling.

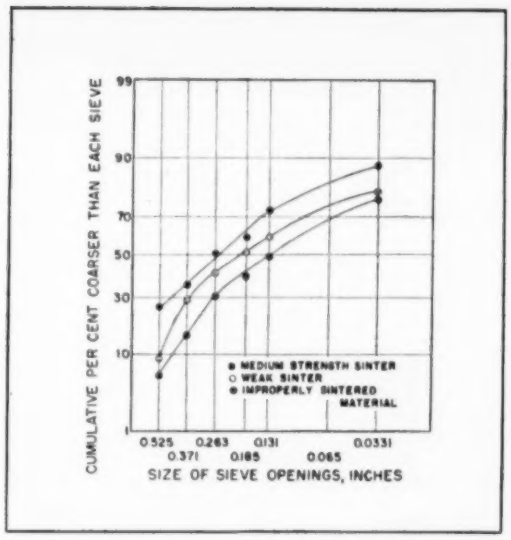

Fig. 4—Values were charted of tumbler testing relatively strong sinter samples, relatively weak sinter samples, and improperly sintered material.

in Fig. 2. Although the sampling method was originally designed for use on a Dwight Lloyd sintering machine, it can also be used on the Greenawalt type of machine. When used on the Greenawalt type machine, the basket is placed on the sintering grate before the charging car passes over it, and is removed just before the pan is dumped.

Testing

The irregular shape and size of the sinter pieces precluded the use of a simple compression test for determining strength; consequently, the shatter test and tumbler test were investigated. In the shatter test, a sample of sinter, approximately 5 lb, is dropped from a hinged-bottom box at a height of 3 ft onto a steel plate. The broken sinter is sieve-analyzed after a specified number of drops. The tumbler test is performed with the use of a standard ASTM coke-tumbling drum. The drum is 3 ft in diam and is equipped with two lifter bars diametrically opposite one another on the inner periphery of the drum. The drum is rotated at a speed of 24 rpm for 200 revolutions, and after tumbling the sample is sieve-analyzed.

To express as single numbers the results of sieve analyses after shattering or tumbling, the method suggested by R. E. Powers[1] was employed. This method involved plotting the size of the sieve openings on a logarithmic scale and the cumulative per cent larger than each sieve on a probability scale as described by J. B. Austin.[2] By interpolating from the plotted data, which in most cases approximated a straight line, the median size could be determined. The median size is taken as the point of intersection of the curve with the 50 pct ordinate.

Results of shatter tests indicated that this test was not severe enough, in that the median size of the sinter after shattering was greatly affected by the starting size of the original sample. On the other hand, the results of tumbler tests revealed that the median size of the sinter after tumbling was almost unaffected by the starting size, and that this test was still selective enough to distinguish clearly between median sizes of sinters of different strength. Fig. 3 illustrates the effect of starting size on tumbler test results. In this plot it can be seen that the starting size had little effect on the median size of the sinter after tumbling, for when the starting size was ½ to ¾ in. and when it was 1 to 2 in. the resulting median size was 0.163 in.

Although the shatter test is not severe enough to eliminate wide discrepancies in median size caused by variations in starting size, the tumbler test is severe enough to reduce such discrepancies to a minimum. Furthermore, the tumbler test is not so severe that it destroys the distinction between sinters of different strength.

To insure getting results that were representative of the full depth of the sintering bed, the entire sample obtained by means of the sampling basket was used as the starting material for the tumbler test. If the sample broke into pieces, all the pieces were used, regardless of their size. Results of tumbling for 10, 20, 50, 100, and 200 revolutions indicated that the median size after tumbling was proportional to the logarithm of the number of tumbler revolutions. After 200 revolutions, little effect on the median size of sinter resulted from one or two additional tumbles. Therefore, the 200-revolution test was selected as a standard test because it tends to minimize differences in the severity of handling the sinter between the sintering machine and the testing laboratory. Fig. 4 illustrates typical values resulting from tumbler testing entire samples procured with the sampling basket. A sample of relatively strong sinter had a median size after tumbling of 0.251 in., a sample of relatively weak sinter, 0.184 in., and a sample taken when the sintering was not progressing satisfactorily, only 0.122 in.

The results of this method of testing were easily reproducible in that adjacent samples from the same pallet gave median-size results that agreed with one another within a few thousandths of an inch.

References

[1] R. E. Powers: Importance of Testing Sinter. Proceedings AIME Blast Furnace, Coke Ovens, and Raw Materials (1951) **10**, pp. 143 to 146.

[2] J. B. Austin: Methods of Representing Distribution of Particle Size. *Industrial and Engineering Chemistry*, Anal. Ed. (1939) **11**, pp. 334 to 339.

Deep Drawing Steel Sheet Surface Quality

by R. J. Walter

ANY discussion of surface quality of aluminum-killed deep drawing steel sheets should undoubtedly be prefaced by the statement that complete control of this quality does not lie in the hands of the sheet producer. High surface quality is simply good matching of the surface requirements of the end use with the actual surface condition of the sheet. The end use is the yardstick by which the surface quality is measured.

Steelmaking and processing practices required to produce a good surface on aluminum-killed deep drawing steel sheets hold certain analogies with those required to produce rimmed steel sheets with good surfaces. These are, of course, good workmanship, proper selection of raw materials, and the use of well maintained equipment.

Basically, the open hearth furnace practice for this grade is the same as for any good grade rimming steel of 0.06 pct max carbon specification, with low residual phosphorous and sulphur. One slight change would be the tapping of the heat a little on the hot side. The addition of aluminum to reach the desired aluminum content can be all to the ladle, or it can be split with part added to the ladle in the form of bar and the balance added to the molds in the form of either shavings or shot.

Pouring is Critical

Pouring of the heat into the molds is a critical point if good surface is to be obtained. The use of clean, warm, and well tarred molds is a must in this practice. Attention also should be paid to the correct setting of the stopper rod to permit proper opening and closing of the stream. Once the nozzle has been centered over the mold the opening up should be such that a pad has been formed on the stool before a full stream is achieved. This opening up is best done with a sweeping motion that is performed in a deliberate but unhurried manner.

Soaking pit practice for this grade should produce a well soaked ingot with a heavy, though loose, scale jacket that is easily removable during the slabbing operation. The avoidance of heavy passes during reduction of the slab in the slabbing mill will minimize any tearing of the slabs. Good cleaning sprays should prevent the rolling-in of any tight soaking pit scale which might be present. Slabs of the aluminum-killed grades are usually scarfed. This scarfing operation is also a critical point in the development of good surface.

Reheating of the slabs requires good temperature, thorough soaking, and a heavy loose scale formation. Complete removal of this scale during rolling is a function of the proper type of cleaning sprays operating at sufficient pressure to remove not only any scale that might have been rolled in but also any secondary scale which has formed during the reduction to that point.

Slivers

Probably the defects of greatest concern with aluminum-killed steels are in a group commonly referred to as *slivers*. This group covers several defects including true slivers, skin pipe or laminations, and scale streaks, which because of the similarity of their appearance are usually designated by the general term slivers since definite identification is usually rather difficult and at times impossible. True slivers can be either mechanical slivers or steel slivers. The mechanical sliver is the result of a gouge or scratch during rolling, and is minimized simply by proper maintenance of the rolling equipment and by a good rolling practice. The steel slivers usually originate from tearing of the steel during the slabbing operation or with cracks developing in the ingot during solidification, therefore rough treatment of the ingots from the time the ingot has been poured to the time it has been charged into the soaking pits should be avoided as much as possible. Skin pipe is a phenomena that is often found in killed steels, or in any steel where the deoxidizing agent is added in amounts in excess of that required for deoxidation. The use of a good mold coat properly applied at the correct mold temperature, and a pouring temperature slightly in excess of that used for rimmed steels is very effective in reducing this type of defect. Rolled-in scale usually goes hand in hand with true slivers and skin laminations. This relation is attributed to the tendency for slivers and skin laminations to open up to the point where scale can form under them and be covered up by further rolling. Removal of this condition is best accomplished by producing a heavy scale in the slab reheating furnaces, good hydraulic cleaning, and an efficient pickling line incorporating a well designed scale breaker, rinsing tanks and sprays, and sufficient control over the pickle liquor to insure best pickling conditions.

Another defect common to aluminum-killed steels is the scab. Minimizing of this defect is also, to a great extent, dependent upon proper application of a good mold coat and pouring at the right temperature. As related to scabs the pouring rate and opening up on the nozzle in the manner that has been previously described are important in avoiding scabs. The proper opening up reduces the number of splash scabs formed. A fast rate of pour results in remelting of some of the scabs that have been formed during the opening up period.

Inspection Vital Factor

Thus, there are a considerable number of factors that are reflected in the surface condition of the sheets but the degree of quality produced has not yet been determined when the sheets arrive at the inspection bench. Here the quality is determined, since at this point the surface condition is measured in terms of surface requirements. This measurement is the most critical operation throughout the entire practice and is controlled entirely by the inspector at this point. This inspector through previous experience and information supplied by the quality control people must, in his own judgment, effectively insure that the surface condition of the outgoing sheets is equal to the surface requirements.

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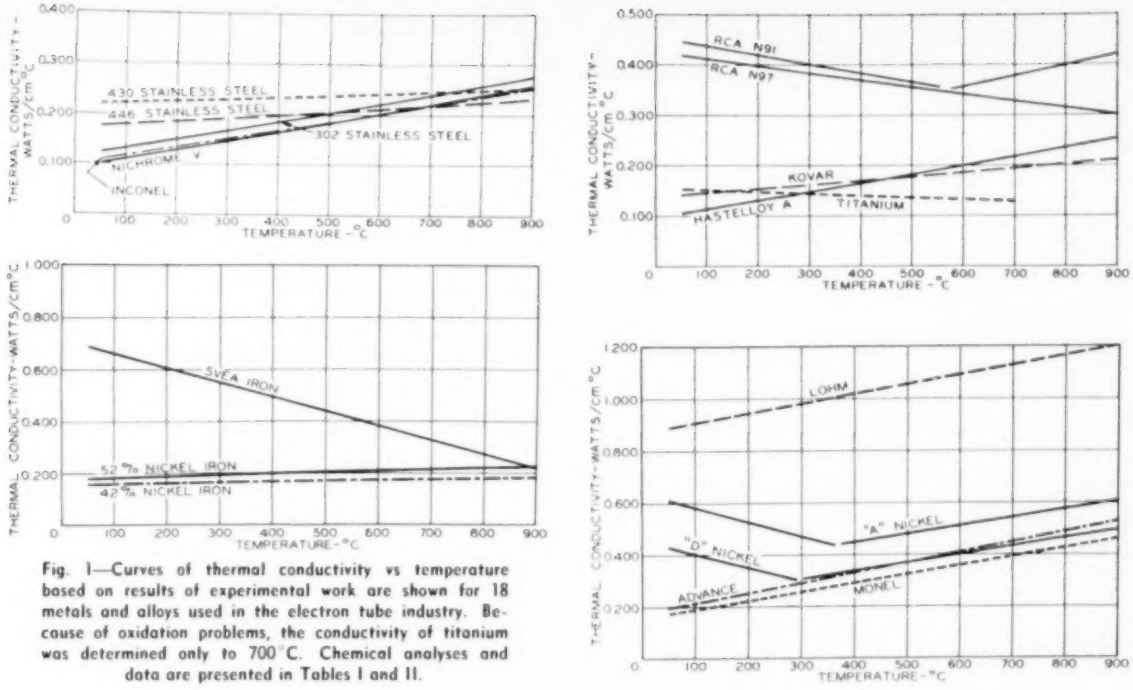

Fig. 1—Curves of thermal conductivity vs temperature based on results of experimental work are shown for 18 metals and alloys used in the electron tube industry. Because of oxidation problems, the conductivity of titanium was determined only to 700°C. Chemical analyses and data are presented in Tables I and II.

Thermal Conductivity Data Presented For Various Metals and Alloys Up to 900°C

by L. Silverman

THERMAL conductivities between room temperature and 900°C have been determined for variout metals and alloys used in the electron tube industry. The method used for the determination of thermal conductivities was a somewhat modified version of the comparative method of Van Dusen and Shelton.[1] This was chosen in preference to those used by Smith[2] and Powell[3] because it did not involve absolute determination of quantities of heat.

In this method, two cylindrical bars of equal diameter, one having known conductivity, are silver soldered end to end. One end of the system is then heated and the other end cooled. Lateral heat loss from the bars is prevented by a surrounding guard tube having heaters and a cooling unit so arranged that a lateral heat balance can be established. This apparatus, has been completely described by Van Dusen and Shelton.[1]

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When a heat balance has been established, Fourier's law can be applied. The fundamental differential equation for heat transfer by conduction is $\frac{dQ}{d\theta} = -KA\frac{dt}{dx}$, where $dQ/d\theta$ is the rate of heat flow in watts, A is the area at right angles to the direction of heat flow in square centimeters, dt/dx is the temperature gradient in degrees Centigrade per centimeter and K is the thermal conductivity in watts for a cross-sectional area of 1 cm^2, a temperature rise of 1°C, and a length of 1 cm.

Because a comparative method is used, the left side of the equation can be ignored. A can also be eliminated because the bar having known conductivity and that having unknown conductivity are of equal cross-section. Only the thermal conductivities and the temperature gradients, therefore, need be considered. The temperature gradients can be calculated from temperature measurements along the bars. The thermal conductivity at any point in

Table I. Chemical Analyses of Materials Tested

Material	Anneal Temp. °C	Analysis, Pct												
		Ni	Fe	Cu	Cr	Co	Mo	Mn	C	Si	P	S	Mg	Ti
Lohm	900	6.05		93.4				0.01		0.01				
***A* Nickel**	900	Bal	0.26	0.063		0.746	0.705				0.036			
Advance	900	44.04		54.79				1.20	0.035	0.003				
***D* Nickel**	900	92.79	1.35			1.27		4.35	0.158	0.06				
Monel	950	66.2	1.88	30.0		0.407		0.919	0.40	0.135			0.032	
RCA N91	1000	78.1				21.6		0.185	0.115				0.01	
RCA N97	1000	59.5				40.0		0.175	0.132	0.19			0.01	
302 Stainless	1100	9.60	71.60		18.40				0.116	0.130	0.021	0.013		
Nichrome V	950	77.94	0.036		19.87			0.06		1.44				
Hastelloy A	1150	57.1	21.4				19.0	2.5	0.072					
430 Stainless	050		82.4		17.2			0.254	0.102			0.035		
Inconel	1050	78.13	6.33		13.94	0.30		0.32		0.33				
446 Stainless	900		76.44		23.58			0.043	0.152			0.021		
52 Pct Nickel Iron	950	50.85	48.5					0.12	0.024			0.003		
Svea Iron	900		Bal					0.02	0.026	0.028	0.021	0.011		
Kovar	900	28.75	53.7			17.15		0.47	0.017					
42 Pct Nickel Iron	950	43.91	55.8					0.22	0.050			0.003		
Titanium	700		0.04					0.10	0.035				0.01	Bal

either bar is inversely proportional to the temperature gradient at that point. Because the absolute thermal conductivity of one bar is known at one point within the working range, the conductivity of the other bar can be determined at all points at which the temperature gradient has been calculated. The temperature gradient at the mean temperature between two adjacent thermocouple junctions is equal to the ratio of the temperature difference between the junctions and the distance between them. No appreciable error is introduced when finite differences are used because the curve of temperature as a function of distance along the length of the bars has in all cases only a small curvature. If a value for conductivity is established for some temperature within the working range, then conductivity values at other temperatures can be calculated from the relationship: $K\frac{dT}{dx} = K\frac{T}{x} =$ constant, where K = thermal conductivity and $\frac{dT}{dx} = \frac{T}{x}$ = temperature gradient.

The above relationships are true for two materials of the same diameter placed end to end as well as for a sample made from one material.

The conductivity values given are relative; because the absolute conductivity for one material is known, however, values for other materials can be calculated. For this experimental work, lead was chosen as a primary standard because the absolute value of the thermal conductivity of lead at 0°C is generally accepted. Because of lead's low melting point, however, *Advance* (55 copper-45 nickel) was used as a working standard.

Experimental Results

A chemical analysis of the materials used in these experiments is given in Table I. The results of the experimental work are given in Table II and Fig. 1. The curves for *A* nickel, *D* nickel, and RCA N91 show abrupt changes in slope at the *Curie* point. The magnetic transformation point of *A* nickel concurs with the generally accepted value. The point shown for *D* nickel, however, is about 50°C below the value usually published. However, Shaw[4] agrees closely with this value. The *Curie* point established for 20 pct cobalt nickel also agrees with that found by Shaw. He also gives a *Curie* point at 740°C for 40 pct cobalt nickel, but this point did not appear in the thermal conductivity determination described here. The curve for 40 pct cobalt nickel is a straight line having decreasing conductivity with increasing temperature.

The curves for the metals iron and titanium also show a decrease in conductivity with increasing temperature. All the other alloys tested have curves which show increasing conductivity with increase in temperatures. Because of oxidation problems the conductivity of titanium was determined to only 700°C.

Errors and Deviations

Van Dusen and Shelton have discussed the possible sources of error: Thermocouple calibration, heat conduction along the thermocouple wires, heat leakage from sample to guard tube, and general temperature drift. Proper precautions make it possible to hold the sum of these errors to a value of ±2 pct.

The *trueness* of the curves determined from the experimental data was checked by the determination of the average deviation by the *least-squares* method. The maximum average deviation for any curve is less than 3 pct.

Table II. Thermal Conductivity of Various Materials, Watts/cm°C

Material	50°C	100°C	200°C	300°C	400°C	500°C	600°C	700°C	800°C	900°C
Lohm	0.889	0.907	0.944	0.980	1.016	1.053	1.089	1.126	1.162	1.199
***A* Nickel**	0.610	0.582	0.525	0.470	0.446	0.447	0.509	0.540	0.587	0.604
Advance	0.194	0.212	0.251	0.289	0.328	0.367	0.405	0.463	0.481	0.520
***D* Nickel**	0.430	0.403	0.350	0.300	0.334	0.367	0.399	0.431	0.464	0.497
Monel	0.173	0.189	0.222	0.255	0.288	0.321	0.355	0.388	0.422	0.455
RCA N91	0.446	0.437	0.419	0.401	0.383	0.366	0.359	0.380	0.401	0.421
RCA N97	0.418	0.411	0.398	0.384	0.370	0.356	0.343	0.339	0.315	0.301
302 Stainless	0.123	0.133	0.151	0.168	0.186	0.204	0.221	0.239	0.256	0.274
Nichrome V	0.103	0.112	0.130	0.148	0.166	0.183	0.201	0.220	0.238	0.255
Hastelloy A	0.105	0.114	0.131	0.148	0.166	0.183	0.201	0.218	0.236	0.253
430 Stainless	0.220	0.222	0.226	0.229	0.233	0.237	0.240	0.244	0.248	0.252
Inconel	0.111	0.119	0.136	0.152	0.169	0.185	0.202	0.218	0.235	0.251
446 Stainless	0.176	0.180	0.186	0.192	0.198	0.204	0.210	0.216	0.222	0.228
52 pct Nickel Iron	0.181	0.183	0.189	0.194	0.199	0.204	0.209	0.214	0.219	0.224
Svea Iron	0.690	0.662	0.606	0.552	0.494	0.438	0.384	0.330	0.274	0.220
Kovar	0.142	0.147	0.150	0.156	0.166	0.175	0.184	0.193	0.201	0.210
42 pct Nickel Iron	0.159	0.161	0.165	0.169	0.171	0.174	0.178	0.181	0.184	0.188
Titanium	0.154	0.152	0.148	0.144	0.140	0.137	0.133	0.129		

Operations at New Cornelia Copper Smelter Of Phelps Dodge Corporation

by James W. Byrkit

Design features and operating methods at the new Ajo smelter are described in detail. Successful operation of a novel method of handling and charging wet concentrates to a deep bath type reverberatory furnace contribute to the daily production of 200 tons of anodes with good results from the standpoint of both metallurgy and economy.

THE New Cornelia Branch of the Phelps Dodge Corp. is located at Ajo, Ariz. Large scale mining operations were started at Ajo in 1917, when a 5000 ton leaching plant was put in service to treat the copper carbonate ore that overlaid the sulphide orebody. In 1924 a 5000 ton flotation plant was built to treat the sulphide ore and the leaching operation was abandoned a few years later. Changes in practice and additions to plant facilities have resulted in a gradual increase in the milling rate which now approaches 30,000 tons daily.

Prior to the erection of the Ajo smelter, flotation concentrate was shipped by rail to Douglas, Ariz., where it was treated in the Phelps Dodge smelter. Construction of a one reverberatory furnace smelter to produce an average of about 200 tons per day of copper anodes at Ajo was started early in 1949. Initial heating of the reverberatory furnace was begun on June 21, 1950. Charging the furnace was started on July 8, and the first anodes were cast on July 14. Neither the anode furnace nor the reverberatory furnace has cooled since the initial firing. Fig. 1 is a photograph of the new smelter.

Built primarily to eliminate the 300 mile haul of concentrate to the Douglas plant, the new smelter was designed to treat the Ajo concentrate, with a minimum of capital investment. A novel, and relatively simple, method of concentrate handling and furnace charging made unnecessary the installation and operation of expensive storage facilities and reclaiming equipment, and obviated the necessity of holding in storage large quantities of copper-bearing materials. (Patents are pending in the United States and abroad on the Ajo process.) In the selection and arrangement of equipment, consideration was given to the full utilization of all smelting facilities, reducing the amount of idle equipment to a minimum. The layout of the smelter is shown in Fig. 2. The

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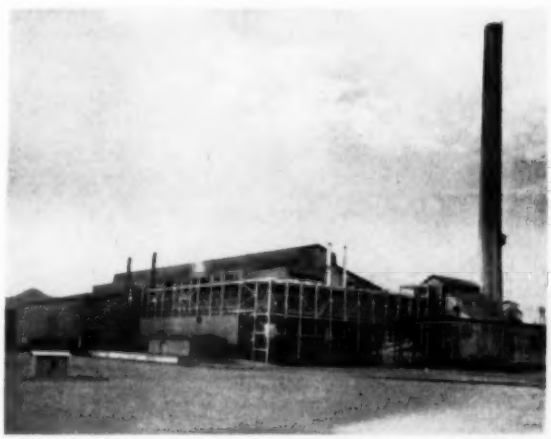

Fig. 1—The new copper smelter, designed to treat the Ajo concentrates, eliminated the shipping of flotation concentrates to the Phelps Dodge Smelter at Douglas, Ariz.

important problem of internal transportation was minimized by arranging the reverberatory furnace, converters, and anode furnace in a compact group and providing storage bins inside the smelter building, within reach of overhead cranes, for fluxing materials and other supplies needed in the daily operation. This can be seen in the sectional view of the smelter given in Fig. 3. Incorporated in the design were many innovations in smelting equipment intended to facilitate operations and effect economies in maintenance expense and manpower requirements. Table I gives operating data of the entire plant.

Metallurgy

The metallurgical practice at Ajo is based essentially upon the use of a single, deep bath reverberatory furnace for smelting wet concentrate, without concentrate storage facilities. The large reservoir of slag and matte maintained in the furnace serves to equalize, to some extent, the day to day variations in the nature and grade of concentrate and permits

The Transactions papers appearing in the Journal of Metals during 1953 will subsequently appear in AIME Transactions Volume 197, and may be permanently referenced to that volume.

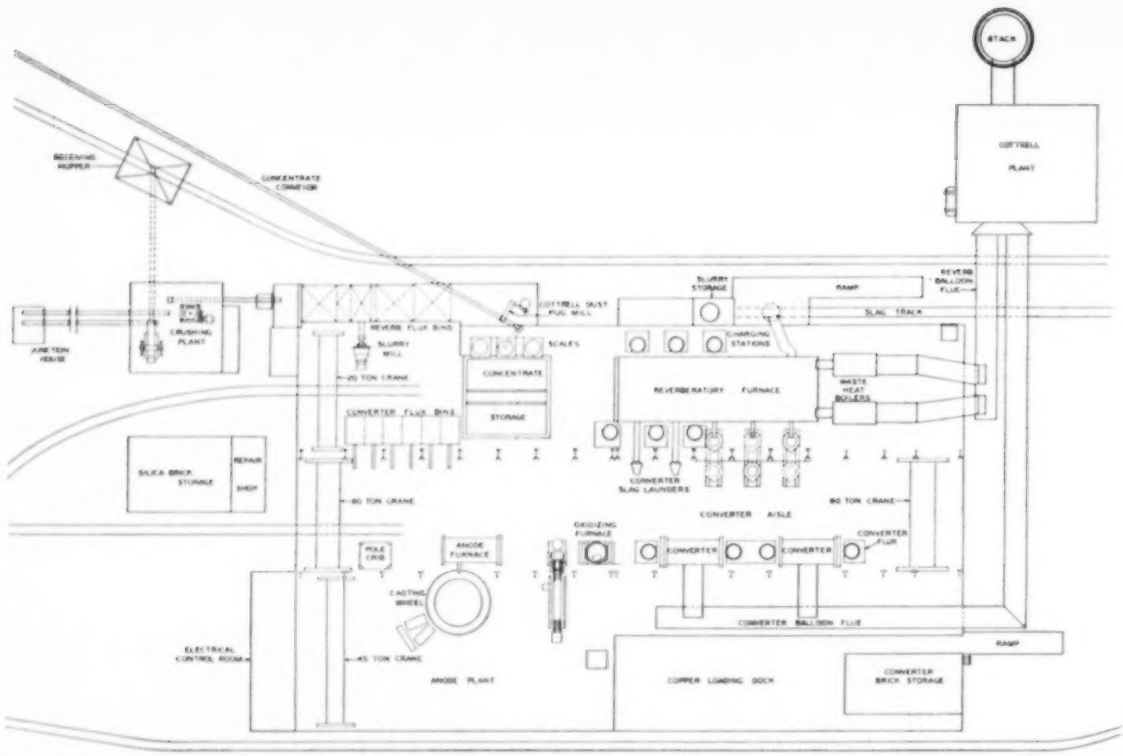

Fig. 2—Arrangement of equipment can be seen in this layout drawing of the smelter. The design permits full utilization of all smelting facilities, and the amount of idle equipment was kept to a minimum.

utilization of converting and refining facilities to the best advantage. The smelter flowsheet is presented in Fig. 4.

Concentrate is taken from flotation plant filters and transported to the smelter on a conveyor belt. Fine crushed limestone is added to the concentrate on the conveyor at a predetermined rate. Ordinarily no other fluxing material is used on the reverberatory furnace charge, which averages 95 pct concentrate. All siliceous fluxing material is charged to converters. The bulk of converter flux is tailing from the old leaching operation. This tailing is about −⅜ in. in size and contains approximately 67 pct silica and 14 pct alumina. Barren quartz from a local quarry is crushed and blended with tailing to maintain a silica-alumina ratio of about 6 to 1. Custom shipments of siliceous ore constitute a minor portion of converter flux. All converter reverts are resmelted directly in converters.

Copper in the Ajo concentrate is present mainly as chalcopyrite, with some bornite and occasional small amounts of chalcocite and native copper. The grade of concentrate varies from day to day, over a considerable range, with accompanying changes in

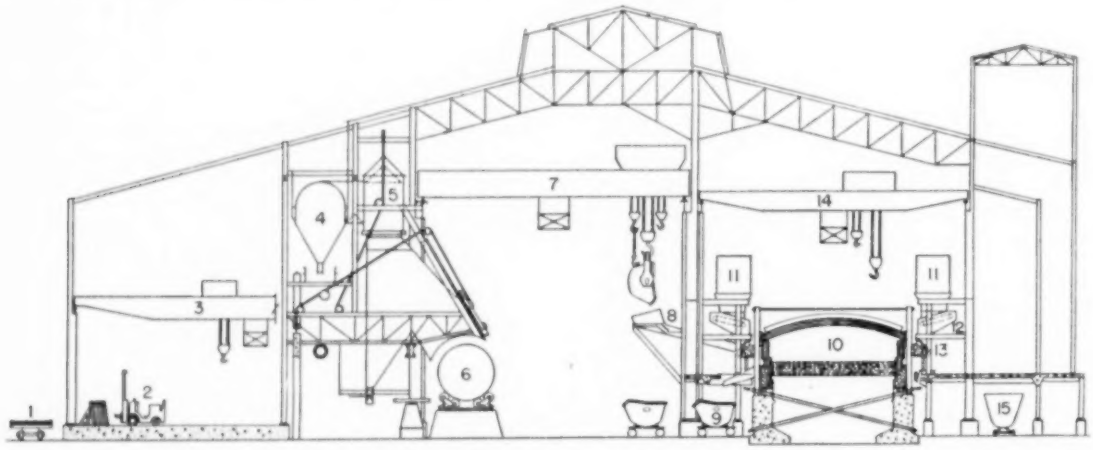

Fig. 3—This sectional view of the smelter shows the positioning of service equipment intended to facilitate internal transportation and operations. 1—Railroad flat car to refinery. 2—Fork lift truck for loading anodes. 3—Anode handling crane, one, 15 ton. 4—Balloon flue, converter. 5—Jug damper. 6—Converters, two. 7—Converter cranes, two, 60 ton. 8—Converter slag return launder. 9—Matte ladles. 10—Reverberatory furnace. 11—Concentrate "can." 12—Concentrate feeder belt. 13—Concentrate slinger. 14—Reverberatory cranes, two, 20 ton. 15—Slag pots.

iron and insoluble material content, depending upon the grade and nature of the mill feed. Matte grade varies with the copper content of the concentrate, ranging from 25 to 35 pct, averaging about 30 pct. Copper content of matte is usually 7.0 to 8.0 units higher than that of concentrate. Slags are fairly uniform in character and composition, copper content varying with matte grade and to some extent influenced by iron carried out in slag.

In the absence of intermediate storage, concentrate is smelted before analyses are available but lack of analyses before charging to the furnace has not been a handicap. Minor adjustments in converter fluxing are made on the basis of analyses of concentrate and final slag for the previous day. When visual appearance of the bath indicates the need, small additions of limestone or silica are made directly to the furnace. Charging of all siliceous flux to converters causes the formation of high-silica slags, but this practice has not perceptibly affected converter linings and has reduced the amount of magnetite returned to the furnace. Furthermore, smelting in converters of practically all silica flux used results in increased furnace capacity for concentrates, which at Ajo is an important factor.

Crushing Plant and Flux Storage

The smelter crushing plant was designed to prepare fluxing materials for converters and reverberatory furnace and to crush reverts and other materials used in the smelter operation. Coarse materials, such as barren quartz and custom ores, are received in either side or bottom dump railroad cars and are dumped into a 75-ton capacity bottom discharge concrete receiving bin. Materials are fed from the bin by pan feeder and belt conveyor into a 36x48 in. jaw crusher, the 3-in. product of which is conveyed to a 4x8 ft mechanically vibrated screen. The screen oversize falls into a 4 ft standard cone crusher set at about ½ in. and remains in closed circuit until reduced to screen undersize. The screen undersize is transported by conveyor and bucket elevator to storage bins.

The leached tailing siliceous flux is loaded by shovel from the dump, which is about one mile from the smelter, into railroad cars and is passed through the plant without crushing. The desired blending of converter flux is accomplished by adding crushed barren quartz in the correct proportion. Limestone, received from an outside source as screened product, likewise requires no crushing. An auxiliary screen is provided inside the smelter building, over which barren quartz is passed, the oversize being used for converter flux, while the 6-mesh undersize is stored for use as furnace patching material.

Storage bins for both reverberatory and converter fluxes are so located in the smelter building that materials may be withdrawn through chutes into boats or containers for handling with overhead cranes.

Table I. Operating Data Total Plant

	Average per Mo. Tons N.M.B.M.* Treated	Pct. Copper Input, N.M.B.M.	Actual Production Average Month Salable Copper	Power Used Kw-Hr Per Ton, N.M.B.M.
1950 (6 mos.)	15,810	28.89	8,692,161	15.79
1951	21,384	25.21	10,541,756	11.21
1952 (6 mos.)	22,822	24.89	11,133,730	11.44

* New metal-bearing material.

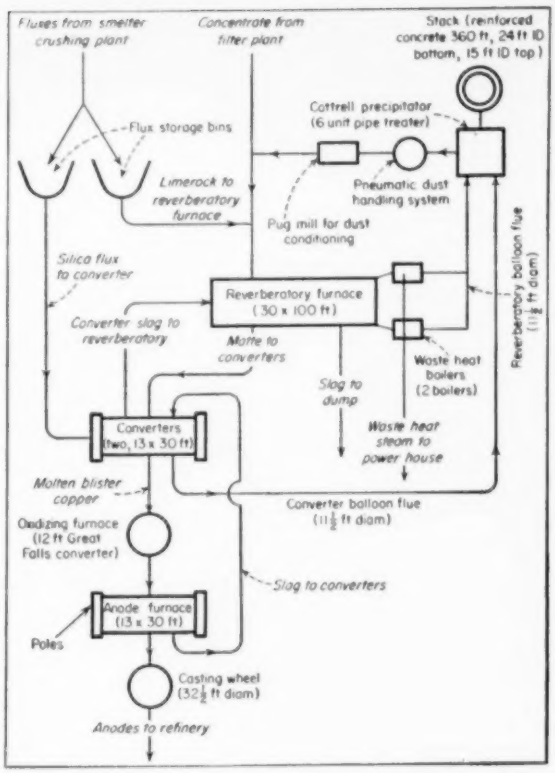
Fluxes from smelter crushing plant
Concentrate from filter plant
Stack (reinforced concrete 360 ft, 24 ft ID bottom, 15 ft ID top)
Cottrell precipitator (6 unit pipe treater)
Flux storage bins
Limerock to reverberatory furnace
Pneumatic dust handling system
Pug mill for dust conditioning
Silica flux to converter
Reverberatory furnace (30 x 100 ft)
Converter slag to reverberatory
Matte to converters
Waste heat boilers (2 boilers)
Slag to dump
Converters (two, 13 x 30 ft)
Waste heat steam to power house
Molten blister copper
Converter balloon flue (11½ ft diam)
Oxidizing furnace (12 ft Great Falls converter)
Slag to converters
Anode furnace (13 x 30 ft)
Poles
Casting wheel (32½ ft diam)
Anodes to refinery

Fig. 4—The processing of the concentrate from the filter plant is diagrammed in the smelter flowsheet. The only fluxing material added to the reverberatory charge is limestone; all siliceous flux is charged to converters.

Reverberatory Furnace

Design and Construction: The design and construction of the reverberatory furnace embodies several unusual features. The furnace, 100 ft long and 30 ft wide inside the brick walls, is mounted on a heavily reinforced monolithic concrete foundation which rests on a slab of concrete 5 ft thick. The walls of the foundation are 4½ ft thick, with footings 8½ ft wide that extend 2 ft below plant floor level. The top of the foundation is 11 ft above floor level. The foundation, as well as the furnace itself, is isolated on all sides, thereby avoiding transmission of expansion forces to the smelter structure that surrounds it.

The construction of a 10 ft subbase for the furnace hearth was accomplished by tamping in 3-in. layers of crushed diorite rock from the open pit, mixed with 20 pct mill tailings to fill voids. Above the subbase a 4½ ft layer of granular magnetite, a product removed from the tailing of a New Mexico copper concentrator by magnetic separation, was tamped to dense compaction, followed by an 8 in. layer of material composed of 90 pct crushed magnetite slag and 10 pct granular magnetite. A 4 in. veneer of fire clay brick was laid on a corbel inside the foundation wall, for protection of the upper 30 in. of concrete against furnace heat.

The details of sidewall and furnace arch brick work are shown in Fig. 5. The basic crucible was tamped in 10-ft sections, using a mixture of granular periclase with minor amounts of ground periclase

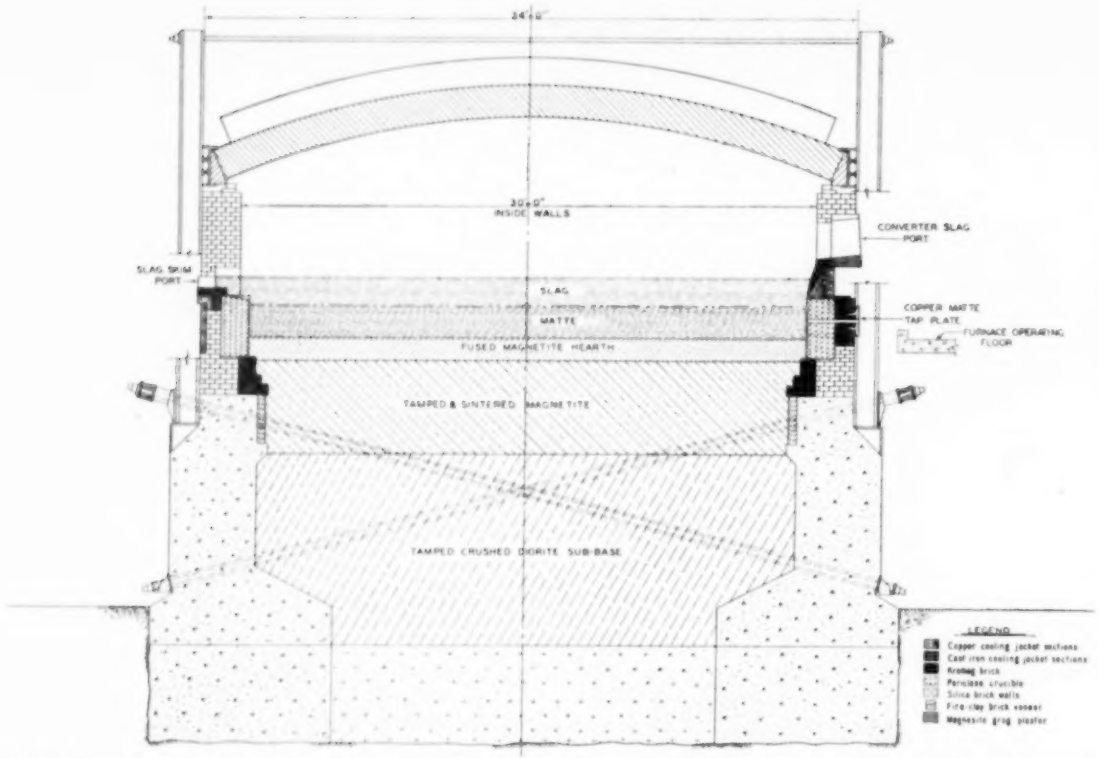

Fig. 5—Sidewall and furnace arch brickwork details, as well as the novel method of buckstay retention, are shown in this composite cross section of the reverberatory furnace. Furnace binding is of unusually heavy construction.

and fire clay, wetted with a controlled amount of sulphuric acid. Immediately above the crucible a row of copper water jackets, 20 in. high, replaces silica brick in the sidewalls extending as far as the skimming bay on one side and through the last matte tap hole on the tapping side. The 20-in. silica brick sprung arch has a span of 31½ ft between skewbacks. The 15-in. ribs, spaced alternately 4 and 6 ft apart, permit installation of a relieving arch through two-thirds of the furnace length, if needed. Above the verb arch, uptake walls are of silica tile sectionally supported construction, and the suspended uptake arch is silica tile.

Furnace binding is of unusually heavy construction. Lateral buckstays are heavy H-beams, resting on an offset in the concrete foundation at the bottom and joined at the top to the opposite buckstay with a 2¼ in. square tie rod, threaded on both ends. Bottom thrust is taken by a heavy channel and angle box beam, running the length of the furnace, which is retained by spring loaded 2-in. tie rods extending diagonally through the furnace subbase to the bottom of the concrete foundation on the opposite side. This novel and effective method of buckstay retention is shown in Fig. 5. End binding is retained by heavy buckstays at each corner, which are tied at the bottom by 3½-in. rods anchored in the concrete foundation and at the top by 3½-in. tie rods running the length of the furnace, all of which are heavily spring loaded.

The magnetite bottom was fused in with an oil fire by regulating the admission of secondary air along the furnace walls and increasing the fuel until the entire surface had become liquid. Additional magnetite tailing was then charged through the sidewalls until a liquid pool had been formed, the surface of which was at the desired final bottom level, about 11 in. below the center line of the matte tap holes. Firing was reduced and the magnetite bottom was allowed to solidify. At this point, molten converter slag, prepared by melting and converting concentrate in converters, was charged through converter slag launders and spread over the chilled bottom. Charging of concentrate was started at once.

Reverberatory Operations: Concentrate is received continuously during operating periods from the mill filter plant over a 24-in. conveyor belt, to which a predetermined amount of crushed limestone is added. Mixed concentrate and limestone are discharged into containers, called cans, resting on a platform scale by means of which the can and contents are weighed and the weights tape recorded. The can resembles a concrete bucket, with heavy lifting bails and sloping inner walls, at the bottom of which are double, air-operated arc gates. The capacity of a can is approximately 10 tons of charge, as-received. When the can has been filled, it is removed from the scale platform by a 20-ton overhead crane, and either set aside until needed, or transported at once to one of the six charging stations. When the charge can is in place over the charging station the bottom gates are opened and the charge fills the small hopper immediately below. A 1200-ton surge bin inside the smelter building is used to equalize mill production against smelter consumption. Concentrate delivered to this surge storage is reclaimed as needed with clamshell and overhead crane.

When the operator is ready to charge the furnace, he raises the door covering the charge port in the furnace sidewall and starts the short feeder conveyor under the charge hopper. Concentrate from the feeder falls into the feed hopper of the charging machine,

locally called the slinger. This machine is a short, high-speed portable belt conveyor pivoted on a vertical shaft that permits lateral swinging. The concentrate falls through the feed hopper onto the rapidly moving belt and, as it moves over the head pulley, is discharged into the furnace. This is illustrated in Fig. 6. The trajectory of the concentrate stream may be varied by elevating or depressing the head pulley, by means of an adjusting device, or by regulating the speed of the belt which is driven by a variable speed dc motor. The average charging rate is about 1 ton per min and the usual charge is 2 to 4 tons. Selection of charging station and timing of the charging period is largely a matter of judgment on the part of the operator, who attempts to keep the surface of the bath covered with a thin layer of unsmelted material. When visual appearance of the bath indicates the necessity, either limestone or silica fluxing material is charged to the furnace directly from cans used for that purpose. As the result of this method of charging small amounts of wet concentrate at frequent intervals, explosions caused by contact of wet charge with molten matte are completely avoided.

The normal liquid bath depth is about 42 in., with 24 to 28 in. of matte. Matte is tapped as required by converters into 175 cu ft ladles resting on electric motored trucks that are energized through the rails with 32 v dc. During the tapping period matte ladles are enclosed in cubicles located under the tap holes below the furnace operating floor which are provided with air-operated hinged doors. Two exhaust fans, one connected to the cubicle and the other to an enclosure around and above the matte launder, effectively remove all fumes from the tapping area, venting the gas to atmosphere outside the smelter building. Matte fall amounts to about 83 pct of concentrates charged.

Slag is skimmed through the furnace sidewall near the verb arch and is laundered into 225 cu ft pots. Fig. 7, a view of the reverberatory from the skimming end, shows the closing of the skimming hole. The usual practice is to skim a train load, consisting of three pots, in succession. Slag is hauled to the dump, about one half mile distant, with 25-ton diesel-electric locomotives. The slag dump is graded with bulldozers and new alignment is made with track shifters. Slag fall amounts to about 75 pct of concentrates smelted.

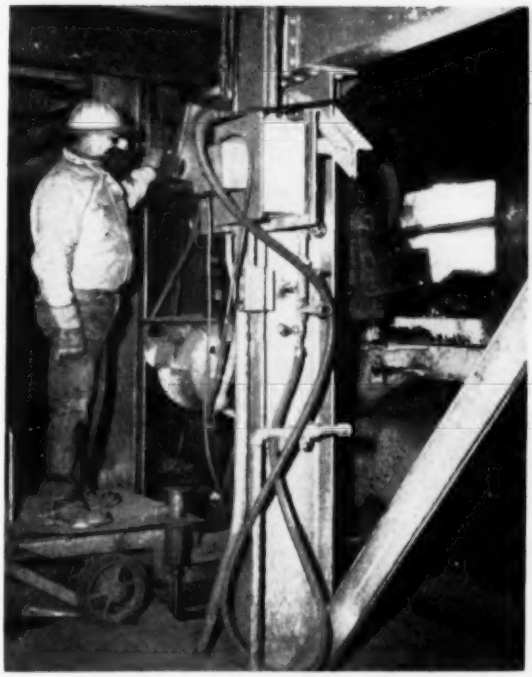

Fig. 6—Concentrate, in 2 to 4 ton charges, falls through a hopper onto a rapidly moving belt and is discharged into the reverb at a rate of about 1 ton per min. The operator regulates charging times and aims at keeping the surface of the bath covered with a thin layer of unsmelted material.

Converter slag is returned to the furnace through duplicate launders in the sidewall near the firing end. Slag inlet ports are formed by heavy, cast copper water jackets set in the brick furnace wall and are equipped with air-operated cast steel gates.

The reverberatory furnace is fired with natural gas. The original installation consisted of multiple jet burners, which functioned well except when a high firing rate was necessary. They were replaced with high pressure, locally made, barrel-type burners. Seven of the barrel-type burners spaced across the burner wall normally burn 185,000 cu ft per hr with good combustion. A view of the furnace from the burner end is shown in Fig. 8. Natural gas is received from New Mexico fields at 300 psi at the plant and

Fig. 7—After slag is skimmed through the furnace sidewall near the verb arch, the skimming hole is closed. Usually three 225 cu ft pots, a train load, are skimmed in succession.

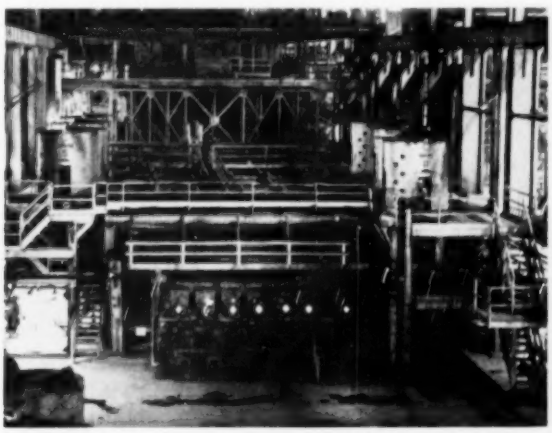

Fig. 8—Seven barrel type burners, firing 185,000 cu ft of natural gas per hr, can be seen in this view of the reverberatory furnace. Standby oil burning equipment is available for emergencies.

Table II. Reverberatory Furnace Data (Dry Ton Basis)

	1950, 6 Mos.	1951	1952, 6 Mos.
Concentrate	91,057	212,298	112,584
Limerock	6,550	12,742	4,987
Silica Flux	5,722	5,387	694
Plant Cleanup	3,085	2,364	960
Solid Charge, total	106,414	232,791	119,225
Furnace Days	175	339	170
Solid Charge per Furnace Day (dry)	608	687	701
Btu per Ton Charge x 1,000,000	6.878	6.674	6.421
Btu per Ton Net x 1,000,000	3.223	3.171	3.002
Converter Slag	51,702	135,103	78,478
Reverberatory Slag	68,857	169,401	89,822
Reverberatory Matte	70,610	168,416	92,140
Steam per ton Concentrate, lb	3,851	3,515	3,422
Waste Heat Recovery, pct	53.14	52.49	53.25

is reduced to 42 psi at the burners. The gas contains about 1100 Btu per cu ft. Standby oil burning equipment is provided for use during periods when the natural gas supply is interrupted. Low pressure oil burners are constructed to fit inside the barrel-type gas burners. Firing can be changed from gas to fuel oil in about 20 min. Furnace combustion is checked frequently by Orsat analysis and a recently installed continuous oxygen analyzer indicates the oxygen content of furnace gases at all times. Effective use of the oxygen analyzer was made possible by the development of a successful method of sampling and cleaning furnace combustion gases by Leonard Klein, research metallurgist. Furnace draft is controlled automatically through dampers in back of each waste heat boiler. An automatically operated butterfly damper was installed in the converter balloon flue to regulate distribution of draft between converter and reverberatory operations. A curtain damper across the furnace at a distance of 10 ft from the verb arch, extends from the furnace roof to about 36 in. above the normal surface of the bath. The damper is constructed of 16 water-cooled panels joined together and covered with silica slurry. The damper serves to stabilize furnace draft, checks flow of dust into boilers, and tends to box in furnace heat.

Furnace sidewalls and arch are maintained by patching with silica slurry, a method developed at Phelps Dodge Corp.'s United Verde branch. Silica is ground and slurried to 65 pct density in a modified Hardinge ball mill and the mill discharge is conveyed through a high pressure blow tank and piping system to an agitator-storage tank located above the furnace operating floor. During patching operations slurry is withdrawn from storage tank into 10 cu ft pressure tanks, to which compressed air is applied, forcing the slurry through hose and pipeline to a nozzle suitable for spraying arch or sidewall. Measurements of arch thickness made twice weekly through 11 measuring holes furnish information as to location in need of patching.

Reverberatory furnace operating data are shown in Tables II and III.

Converters

The general arrangement of the converter aisle is shown in Fig. 2, and a view of the converter aisle is given in Fig. 9. The aisle is 396½ ft long and 63 ft wide. Two cranes, each equipped with a 60-ton main hoist and two 20-ton auxiliary hoists, have a span of 60 ft with crane rails 55 ft above the floor of the aisle. Main hoist dc motors are 135 hp and auxiliary hoist motors are 75 hp. All hoist drives are equipped with two magnetic brakes, one on the armature shaft extension and one on the motor pinion shaft extension. All hoisting ropes are 1 in. preformed plow-steel, with independent wire rope center construction. Crane repair platforms are provided at each end of the aisle, above which are mounted 5-ton repair cranes.

Two 13x30 ft Peirce-Smith converters were moved from the Clarkdale smelter to Ajo over Arizona highways by motor truck. These converters are spaced on 60-ft centers and are opposite the reverberatory matte tapping area. The converter shells are 1¼-in. plate, welded in 10-ft sections. Welded joints are reinforced with heavy butt straps welded to the shell on both sides. The 1-in. end plates are each stiffened with four I-beams and are retained in place with bolts and springs. Riding rings are mounted at the ends, clear of the tuyere section, and are riveted to the shell. Details of brick lining are shown in Fig. 10. Full insulation of converter shell and ends is believed to result in the retention of sufficient heat to resmelt all reverts and relatively large amounts of siliceous material in converters.

Converters are driven by 75-hp ac motors, direct connected to speed reducers and intermediate gears with a heavy pinion engaging a large gear bolted to one of the riding rings. Drive motors are equipped

Table IV. Converter Data (Dry Ton Basis)

	1950, 6 Mos.	1951	1952, 6 Mos.
Matte	70,610	168,416	92,140
Reverts	9,707	24,735	16,550
Barren Quartz	0	6,707	6,060
Leached Tailing	18,721	43,285	22,110
Other Flux	1,717	1,603	2,094
Net Stand Days	161.75	312.60	161.60
Tons Bullion per Stand Day	164.54	202.90	207.34

Table III. Analyses of Reverberatory Operations (Pct)

	Cu	SiO_2	Al_2O_3	Fe	CaO	S	H_2O	Oxygen Ratio	Fe_3O_4
Reverberatory Solid Charge									
1950 (6 mos.)	26.57	12.3	2.7	22.8	3.6	26.2	6.62		
1951	28.02	10.2	2.6	23.3	3.0	27.8	7.62		
1952 (6 mos.)	28.78	7.6	2.2	25.4	2.5	29.3	7.50		
Concentrate Smelted									
1950 (6 mos.)	30.27	8.0	2.5	25.7	0.5	30.3			
1951	30.41	8.5	2.6	25.3	0.3	30.4			
1952 (6 mos.)	30.21	7.3	2.2	26.8	0.2	31.8			
Reverberatory Slag									
1950 (6 mos.)	0.469	37.3	7.5	43.0	5.2	1.0		2.11	
1951	0.478	37.8	7.7	42.9	4.8	0.5		2.17	
1952 (6 mos.)	0.466	37.5	7.0	44.7	4.1	0.7		2.08	
Reverberatory Matte									
1950 (6 mos.)	38.21			33.1		25.2			8.8
1951	38.10			32.4		25.7			7.4
1952 (6 mos.)	37.31			33.3		25.9			6.3

Table V. Analyses of Converter Operations (Pct)

	Cu	SiO_2	Al_2O_3	Fe	S	CaO	Fe_3O_4	MgO
Converter Slag								
1950 (6 mos.)	4.42	22.3	4.5	47.1	1.8	0.7	31.7	
1951	4.74	26.6	4.9	43.5	1.7	0.8	20.4	
1952 (6 mos.)	4.76	29.8	4.7	41.9	1.6	0.6	14.8	
Leached Tailing for Converter Flux								
1950 (6 mos.)		66.7	13.9	4.2		0.8		0.9
1951		67.1	14.7	4.2		0.9		1.4
1952 (6 mos.)		67.2	14.5	4.1		0.9		1.3
Barren Quartz for Converter Flux								
1951		94.1	2.7	1.3		0.9		0.4
1952 (6 mos.)		94.8	1.7	1.3		0.7		0.3
Mixed Converter Flux								
1950 (6 mos.)		68.0	13.2	4.0	0.8	0.9		
1951		70.8	12.6	3.7	0.1	1.0		
1952 (6 mos.)		73.4	11.6	3.4	—	0.9		

with magnetic brakes and are controlled from the operating platform.

Each converter is equipped with 52 ball valve tuyeres, with 2-in. pipes rolled into the seat. The middle 16 tuyere pipes are reduced in diameter by seating a 1½-in. pipe inside the 2-in. tuyere pipe. Converter air is supplied by two 30,000 cu ft steam-driven turboblowers, located in the plant power house. Air is delivered at converters at a normal pressure of 13 psi, which may be varied by a system of controls on the converter operating platform. A warning signal is sounded when pressure drops below normal, giving the operator time to turn the converter out of stack if it is blowing. Air volume to converter will average about 25,000 cu ft per min with 30,000 cu ft per min maximum. Main blast valve is pneumatically operated and is controlled from the skimming platform. Air volume is indicated by a flow meter near the punchers platform and is recorded continuously on a meter in the converter office. Blowing time for each converter is recorded by a timing device in the office.

Converter hoods are of welded steel with cast iron liners. A sliding apron, covering the hood while the converter is blowing, is operated through an air cylinder and heavy wire ropes. A cast steel bell damper in each converter uptake just ahead of the balloon flue is raised and lowered by a motor-driven wire rope hoist. Adjustable punchers platforms are raised and lowered with a hydraulic lift. Punchers platforms and operators platforms for both converters are connected by hand-railed walkways.

Tuyere blocks are 18 in., four piece, hard-burned chrome-magnesite. Tuyere line campaigns will average about five months, finishing 12,000 to 15,000 tons of copper per set. When tuyere repair is made, brick immediately above and below tuyere line is renewed and brick is replaced in arches around the mouth, in the ends, and in the converter bottom when necessary. Steel work around the mouth is overhauled during repair periods and necessary repairs are made to silica feeders, stack, and tuyeres. Cast steel pouring lips are covered with several layers of hard facing and usually last through two or three campaigns.

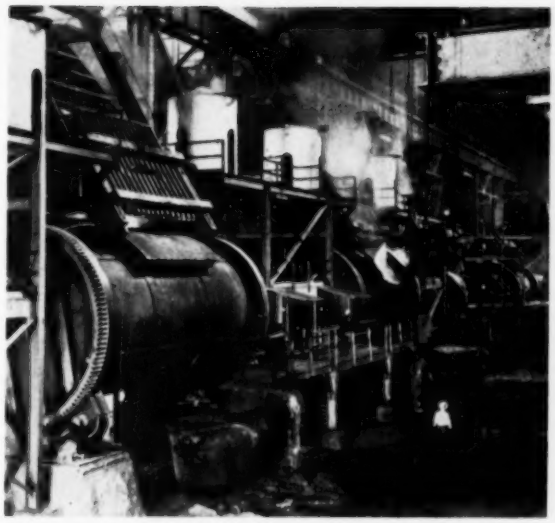

Fig. 9—Two 13x30 ft Peirce-Smith converters, finishing four charges of blister copper daily, are located in the converter aisle.

Normally, converters are charged with four 15-ton ladles of matte which is fluxed, blown, and skimmed, and sufficient additional matte is added to make up a charge of approximately 50 tons of blister copper. Four charges of blister copper are finished daily. Both converters are operated on the day shift and one converter only is blown on the afternoon and night shifts. Converter operating data are given in Tables IV and V.

Mixed converter flux is drawn from storage bins into cans similar to those used for handling reverberatory charge. The cans are transported by overhead crane to a platform above the converters where a charging station is located at either end of each

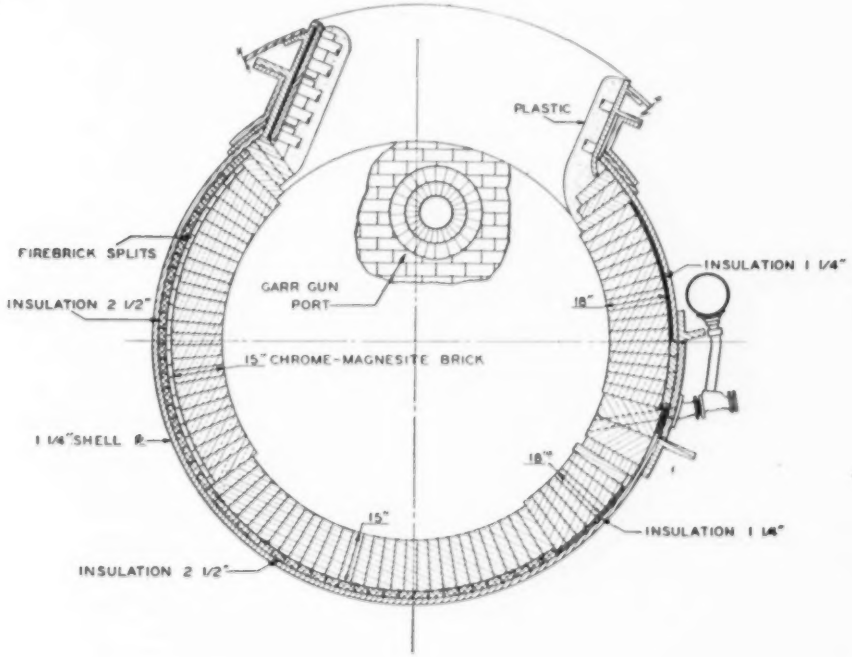

Fig. 10—Sufficient heat is retained by the full insulation of converter shell and ends, as shown in the detailed drawing of the brick lining, to resmelt all reverts.

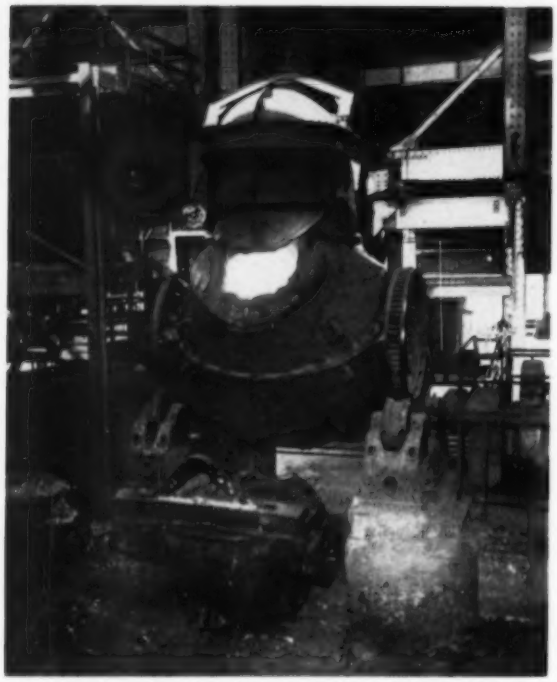

Fig. 11—Blister copper is overblown in this oxidizing and holding vessel, a remodeled 12 ft Great Falls converter. Blowing copper high in this vessel, it is believed, results in longer brick life in the large converters.

converter. The flux flows through the can bottom discharge gates into a hopper which feeds Garr guns at each end of the converter. Additional storage bins make available fluxing materials with high silica content when needed. This method of handling converter flux has been entirely satisfactory and affords considerable flexibility in regulating this phase of the operation.

Converter aisle cleanup is accomplished to a large extent with a small bulldozer. The bulldozer operator cleans the aisle itself and the area under the converters before the start of the day shift, pushing the accumulated slop and spill into a pile at one end of the aisle. Ladle shells are loosened on a copper bumping block, dumped on the revert pile, and broken up with a drop ball. The entire accumulation is loaded into 10-ton boats by crane-operated clamshell and charged to converters as needed. Dust from converter hoppers and from the converter balloon flue is drawn into a dump truck, dumped into a large boat, and charged to converters as operating conditions permit.

Oxidizing Furnace

A departure from conventional converter practice was made at Ajo, sometime after the smelter was started, by overblowing blister copper in a separate oxidizing and holding vessel. An old 12-ft Great Falls converter was slightly remodeled and installed for this purpose, equipped with a close fitting swinging hood and a stack vented to atmosphere, as shown in Fig. 11. The converter is lined with 9-in. chrome-magnesite brick and all but four of the sixteen 2-in. tuyeres are blocked off. The opening in the converter cap was reduced in size and, to make a closer fit with the hood, the normal position of the cap on the shell was reversed. An opening was made on one side of the cap through which a gas burner is inserted for heating the vessel and the copper charge.

When a copper charge is blown to light blister in one of the Peirce-Smith converters, the final slag is dammed off with cold reverts to permit the copper to be withdrawn clean into ladles and transferred to the oxidizing furnace. The charge is blown with converter air at about 13 psi under a fairly heavy gas fire. Complete sulphur elimination in a 50-ton charge is reached in about 30 min. No slag is formed in the oxidizing operation and the overblown copper carries an average of about 5 pct cuprous oxide. Further blowing in the anode furnace is not necessary.

It is believed that blowing copper high in a separate vessel results in longer brick life in the large converters, as it generally has been felt that conditions reached during the overblowing period were destructive to converter lining. The elimination of oxide slag in converter operation is a distinct advantage, as this material is dangerous to handle and slows down the operation while being skimmed and returned to converters. When needed as such, the vessel also may be used as a holding furnace, thereby permitting greater flexibility in the operation of closely designed converting and refining equipment.

Anode Plant

The anode plant is located in and adjacent to the converter aisle, with the single anode furnace in line with the converters. Design and construction of equipment used in both the refining and the casting operations included many innovations which have contributed largely to the effective performance and economical operation of the anode plant. Anode operating data are given in Table VI.

The anode furnace, shown in Fig. 12, is 30 ft long and 13 ft diam. The shell is 1½-in. plate, welded construction. The heads are dished ¾-in. plate with a heavily reinforced ring around the circumference 11 in. wide, making a slip fit inside the shell. The heads are fastened to the riding rings with eight spring-mounted bolts to accommodate expansion. A 4-ft round opening in the shell near one end serves as a charging port for copper and as an outlet for combustion gas; the copper pouring spout is located about 90° below the charging port near the opposite end of the furnace. Unlike most rotating anode furnaces, which have a common opening for poling and release of combustion gas, the poling door of the Ajo furnace is in one end, about 3½ ft above the center line of the furnace. The 30-in. opening is provided with a heavy steel ring bolted to the head, for the protection of brick lining during the poling operation. A hinged door is closed except during the time the furnace is being poled. Small ports in either end of the furnace serve for blowing with high pressure air when necessary, or for gas burners. The anode furnace drive is similar to converter drives, except that direct current is used.

The anode furnace is equipped with a swinging hood that covers the charge opening during the poling operation and is elevated when copper is

Table VI. Anode Copper Data

				Anode Plant Data Per Ton Anode Copper	
	Cu, Pct	S, Pct	O_2, Pct	Poles, Lb	Fuel, Million Btu
1950 (6 mos.)	99.77	0.01	0.156	69.9	1.686
1951	99.75	0.01	0.129	64.3	0.856
1952 (6 mos.)	99.69	0.01	0.130	73.0	1.179

charged. The hood is connected to a steel stack which vents combustion gas to atmosphere through the roof of the building. The furnace is lined with chrome-magnesite brick with a 2½-in. layer of insulating material between the brick lining and the steel shell and ends. Retention of heat in the anode vessel by complete insulation results in better fuel economy as well as improved working conditions on the furnace operating floor.

As sulphur is completely eliminated from the copper charge during the blow in the oxidizing furnace, blowing in the anode furnace is limited to cleaning up the charge and skimming off a minor amount of slag just before the charge is poled.

Oak poles are unloaded from cars into a pole crib at the poling end of the furnace. Poles are raised to the operating floor with an air-operated derrick as needed and are manipulated in the furnace with a small air hoist. A heavy concrete pier immediately below the poling door is provided with rings holding a heavy chain which passes over the pole and takes the strain exerted by the hoist, thereby protecting the poling door and brick lining from damage during the poling operation.

The 700 lb anodes are cast on a 22-mold gear and pinion driven wheel. Copper from the furnace pouring spout is received in a hydraulically operated spoon and poured from it into molds at a normal rate of about 40 tons per hr, as can be seen in Fig. 13. The two dc motors driving the casting wheel are controlled on the Ward-Leonard principle, receiving power from a special dc generator. The armatures of the two 115-v motors are connected in series across the generator terminals.

Control of the casting wheel is semiautomatic. Started by push button, the wheel stops automatically when the next mold is in position under the spoon. The start of motion and acceleration is smooth enough that the wheel can be moved immediately when the mold is filled and cooling of liquid copper with an air blast is not necessary. Cooling water is applied through a series of sprays starting after the surface of an anode has chilled. Lifting pins to raise the top of the anode from the mold are actuated by a lever and roller arrangement that moves over an inclined track under the wheel. Anodes are lifted from molds by pneumatically operated tongs and dropped into one of two bosh tanks, each of which holds 44 anodes. Cooled anodes are lifted from the bosh tanks by a 15-ton overhead crane.

Because of the smooth operation of the casting wheel, rolled edges are seldom formed and little trimming is required. Anodes are carefully inspected and loaded on railroad cars with vertical-lift storage battery trucks. The concrete floor adjacent to the casting wheel, armored with cast iron grids cast into the concrete, affords ample floor space for inspection and loading and an adjoining large concrete platform is available for anode storage when copper shipments are interrupted.

Copper pouring spoons are relined after each pour with a sand-cement mixture and pouring lips are renewed. The newly lined spoon is cured several days under a gas flame and is warmed up to operating temperature before being put into service. Anode molds, weighing about 5000 lb, are cast on the wheel as required, usually before the start of a regular pour. The average life of a mold is about 450 tons of anodes. During the copper pour, molds

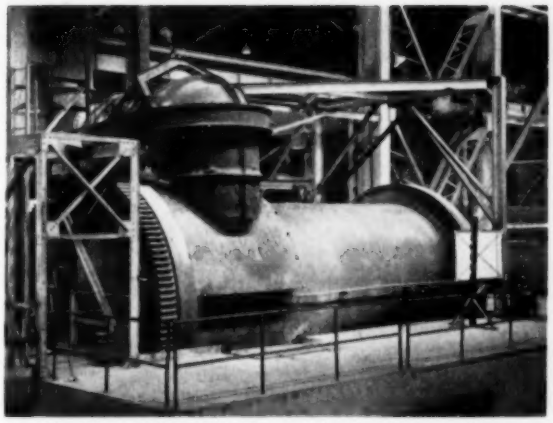

Fig. 12—The rotating anode furnace, located in and adjacent to the converter aisle, has separate openings for poling and release of combustion gas. Blowing in this furnace is limited to cleaning up the charge and skimming off some slag before poling, as all sulphur is eliminated in the oxidizing furnace.

are sprayed while hot with a silica wash to prevent anodes from sticking to the molds.

The operation of the anode plant has been satisfactory in all respects. Repairs to refining and casting equipment have been light and production of high quality anodes has been consistent. Rejection of imperfect anodes has been negligible.

Waste Heat Boilers

Combustion gases from the reverberatory furnace pass through two waste heat boilers, the inlets to which are provided with water-cooled cut off dampers, making it possible to isolate completely either boiler for inspection or repairs. The boilers are not baffled and operate with a draft loss of 0.10 in. of water from inlet to outlet. Design pressure of the boilers is 950 psig, with superheater outlet conditions at 800 psig and 750°F.

Water walls in the furnace section have a heating area of 3730 sq ft per boiler and the boiler section 11,200 sq ft, making a total of 14,930 sq ft of heating area. Normal evaporation is 50,000 lb of water per hr per boiler, with a heat recovery representing 53.0 pct of total fuel used in the reverberatory furnace. Boiler feed water temperature is maintained at 325°F and enters the boiler at 1000 psig.

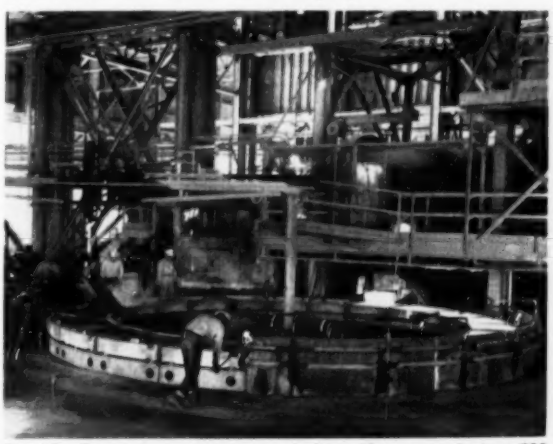

Fig. 13—Copper is cast at a rate of about 40 tons per hr into 700 lb anodes on this 22 mold anode casting wheel. Anode molds are cast on the wheel as required, and have a life of about 450 tons of anodes.

Water wall and slag screen tubes are cleaned with 24 soot blowers, while superheater elements and boiler tubes are equipped with 9 soot blowers in each boiler. Soot blowers are operated automatically from the boiler control panel, using superheated steam at 500 psig. Little hand lancing of boilers is required. Dust trapped in boilers is collected in gated hoppers and is dropped to the ground below where it is moistened with water and loaded into dump trucks. Reclaimed dust is mixed with converter flux in the crushing plant receiving bin and is resmelted in converters.

Flue System, Cottrell and Stack

Converter and reverberatory gases are discharged into separate steel balloon flues, 11½ ft in diameter. The two flues enter a common junction box at the Cottrell inlet, where a baffle system mixes the gases before entering the precipitation chamber. Heavy dust that settles to the bottom of the flues is removed by drag chains, reverberatory dust being discharged to the Cottrell blow tank system, while the converter dust is collected in a hopper from which it is removed by dump truck and returned to the converter aisle for resmelting. Stack and Cottrell temperatures have not as yet indicated the necessity for insulating either of the balloon flues.

Cottrell precipitation consists of six units of pipe treaters, each unit containing 120 12-in. vertical pipes 16 ft long. All units are parallel and down draft. The discharge electrode is a small chain suspended in the center of the 12-in. pipe. All units are hand rapped. Exit gases pass upward around the pipes and then laterally to and through the outlet flue into the chimney. A bypass flue, closed by a jug damper, may be used to pass gases directly into the treater outlet flue uptake, if necessary. Treater walls are acid-proof brick laid with acid-proof mortar.

Power to the treaters is furnished by two mechanical disk rectifiers, each of which is coupled to a 25-kva motor generator set, driven by a 40-hp motor. A third rectifier unit is available as a spare. An exciter furnishes the field for the generator. Two of the four leads from rectifier stationary shoes are connected to terminals of high voltage transformers which have taps ranging from 50,000 to 100,000 v. Normally the 62,500-v tap is used.

Converter gases entering the Cottrell treater vary over a fairly wide range, because of intermittent operation, having an average of 475°F. Reverberatory gas temperature is fairly uniform at about 495°F. Temperature of Cottrell outlet gas averages 430°F. Under present operating conditions, the volume of gas treated is approximately 340,000 cu ft per min, somewhat higher than the normal capacity of the precipitator. However treater efficiency is satisfactory, stack testing showing consistently a loss of less than ¼ of 1 pct of total plant copper input.

Cottrell dust is collected in hoppers, one under each treater unit. The dust is periodically dropped into McLeod blow tanks located under the hoppers and is pneumatically transported from one tank to another in series until it reaches the sixth tank. From the last tank in the series the dust is transported with high pressure air several hundred feet to a cyclone collector, which is equipped with wool filter bags, and is stored in a small bin. Once each shift the dust is withdrawn from the bin, moistened with spray water, put through a horizontal pug mill and delivered to the conveyor belt carrying concentrate into the smelter. Average dust recovery is about 6 tons per day.

The chimney is of tapered reinforced concrete construction, lined with acid-resisting brick laid in acid-proof mortar, and has 2 in. of fiber glass insulation between the concrete and the brick lining. The stack is 360 ft high, 24 ft ID at the bottom and 15 ft at the top. It is protected by lightning rods with copper cables to ground. A testing station is located 127 ft from the ground. At 425°F stack inlet temperature and 60°F outside temperature, total stack draft is 1.70 in. of water. Draft drop through the Cottrell plant is 1.10 in., and the remainder is divided between reverberatory and converter operation by automatically regulating the butterfly damper in the converter balloon flue.

Smelter Organization and Maintenance

Smelter operation is under the control of the general smelter foreman. The reverberatory, converter, and anode departments each have a general foreman, with shift foremen on each of the operating shifts. The crushing plant, mason crew, and Cottrell plant each have a day shift foreman. Testing work is carried on by the smelter engineer.

The reverberatory furnace floor is flushed with water each shift. Grated drainage openings connect with a launder system which discharges all floor cleanings into a concrete sump. A motor-driven pump, actuated by a float switch, delivers the accumulated washings to a concentrate storage tank from whence it is pumped periodically to the mill filter building and joined with current concentrates for return to the smelter. The concrete anode operating and trimming floor, the copper loading dock, and the area behind converters is swept daily with a power sweeper. The area under converters and the converter aisle is cleaned daily with a small bulldozer.

A small repair crew is maintained to take care of running repairs and minor alterations. Major repairs and alterations are handled by the general plant shops which also take care of mine and mill heavy maintenance work.

Summary

After more than two years of entirely successful operation, performance of the Ajo smelter, both from a metallurgical and an economical standpoint, clearly demonstrates that the original objectives have been satisfactorily achieved. Day to day smelting of concentrator output has resulted in a straight flow of copper-bearing material from the mine to finished anodes. Attainment of a relatively high smelting rate through the single reverberatory furnace and a high rate of copper production per man-shift has resulted in an overall cost of producing anodes from concentrates that is favorably comparable with costs achieved in the Corporation's multifurnace copper smelters.

Acknowledgments

The author of the general plan of the Ajo smelter was Charles R. Kuzell, General Manager of Phelps Dodge Corp., who also was responsible for the introduction of numerous innovations in smelting practice and design of smelter equipment. The development of these ideas was carried out by the Western Engineering Dept., under the direction of H. V. Kruse, Chief Mechanical Engineer. Structural design, and the design or selection and arrangement of electrical and auxiliary mechanical equipment and the construction program was under the direction of Wilbur Jurden, Consulting Engineer, New York. Mr. Jurden was represented in the field by Stewart Carpenter, Construction Engineer.

Effect of Dissolved Sulphur on the Surface Tension Of Liquid Copper

by C. F. Baes, Jr. and H. H. Kellogg

The effect of dissolved sulphur on the surface tension of liquid copper has been measured by a sessile-drop method at 1120°C. Sulphur is highly "surface active" in liquid copper; the surface tension lowering is about 650 dynes per cm for a sulphur content of 0.6 pct. The degree of adsorption of sulphur on the liquid copper surface has been calculated, and various models of the surface layer have been tested against the results.

THE science and technology of surface chemistry and physics at room temperature are relatively advanced, and the engineer possesses a background of information that allows him to predict and control many phenomena—such as frothing, emulsification, wetting of solids by liquids, flow through capillaries, adsorption, mineral flotation, penetration of porous bodies by liquids—which are related to the behavior at interfaces. Surface phenomena in high temperature systems are probably of equal importance to the metallurgist and ceramist, but at present are incompletely understood. Frothing of liquid slags, penetration of refractories by liquid metals and slags, infiltration of powder metallurgy "skeletons" by liquid metals, bonding of metals to ceramic oxides in the manufacture of "cermets," wetting of one metal by another for purposes of soldering and welding, are but a few of the many phenomena of metallurgy that are controlled by surface behavior.

Concerning the most elementary of surface properties—the gas-liquid surface tension—the existing data on liquid metals are inadequate. For the surface tension of "pure" metals the results of different investigators differ by 10 or 20 pct. The role of surface-active agents in liquid metals is relatively unknown, and it is not unlikely that the discrepancy in the values for the surface tension of pure metals is due to the presence of traces of surface-tension-lowering impurities in the samples. Based on analogy to room-temperature systems, the highly surface-active impurities in liquid metals should be elements or compounds of limited solubility in the liquid metal and possessing considerably weaker intermolecular bonding forces than the metal itself. The nonmetals of the V, VI, and VII groups of the periodic table might be expected to act in this way.

The experimental work in this paper represents a study of the surface tension of pure liquid copper in hydrogen at 1120°C, and the effect of dissolved sulphur (which is a strongly surface-active agent in liquid copper) on the surface tension of copper. The Gibbs equation has been applied to the results and an attempt made to correlate the surface tension behavior with two models of the interfacial layer.

The sessile-drop method for surface-tension measurement at high temperatures which was used in this investigation has been described by several investigators.[1,2] It involves the measurements of the dimensions of a stationary liquid drop on a horizontal surface. From these dimensions and the liquid density, the surface tension of the liquid may be calculated either by means of various approximate formulas, or more accurately by means of the Bashforth and Adams tables.[3] For liquid drops which exhibit a contact angle greater than 90°, by far the simplest dimensions to measure are those indicated in Fig. 1, i.e., the diameter of the maximum horizontal cross section ($2X$) and the maximum height (Z) above this plane. In order to calculate the contact angle and the volume of the drop, the diameter at the base of the drop ($2a$) and the total height (h) also must be measured.

Upon investigation of the literature, the authors failed to discover a concise statement of the accuracy of the method when used in conjunction with the Bashforth and Adams tables, hence a careful examination of the tables was made and the curve shown in Fig. 2 was constructed. The "error factor," plotted as the ordinate, is the factor by which the

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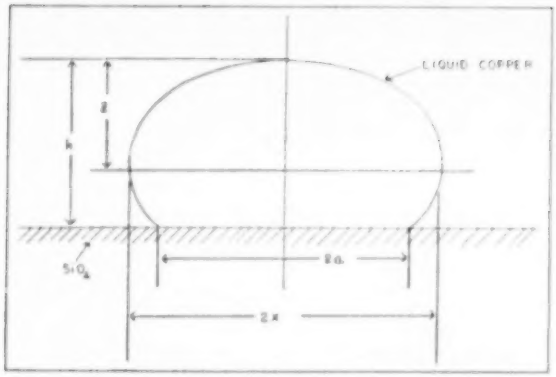

Fig. 1—Diagrammatic representation of a sessile drop of liquid copper on a solid surface.

uncertainty in the experimental measurement X/Z (Fig. 1) is multiplied to obtain the uncertainty in the calculated surface-tension value (assuming no error in the liquid density). The curve clearly indicates the desirability of using drops of the maximum convenient size (thus of the maximum X/Z value).

Experimental

The experimental method involved the following steps: 1—the formation of a sessile drop (of approximately known sulphur content), at 1120°C under hydrogen, in a vacuum-tight furnace tube; 2—photographing the incandescent sessile-drop by means of a fixed-focus camera; 3—measurement of the resulting plates by means of a microcomparater; 4—microanalysis of the solidified drop for sulphur.

Apparatus: The furnace-tube assembly (Fig. 3) consisted of a fused-silica tube (1⅜ in. ID, 1¾ in. OD x 30 in.) with standard taper, 45/50 male joints ground at each end, which provided a vacuum-tight seal when assembled with the water-cooled female joints. The lubricant used in these joints (Apiezon "N" grade stopcock grease) retained its consistency when the furnace was in operation, provided cooling water was passed through the water jackets continuously. The temperature of the central portion of the furnace tube was controlled to ±3°C by means of a Brown temperature-controller activated by a chromel-alumel thermocouple inserted in a silica thermocouple tube. The thermocouple tube contained a silica-pyrex graded seal near the ground joint. The thermocouple was repeatedly checked by noting the temperature reading at which the copper sample began to melt (melting point, 1083°C). The resistance furnace was constructed of a nichrome-wound alumina core packed in insulating brick. The pumping system consisted of a fore pump and a three-stage oil diffusion pump. The all-glass vacuum system, used in conjunction with the vacuum furnace, contained a $CaSO_4$ and a P_2O_5 drying column through which H_2 was introduced, a Töpler pump for regulating gas pressures, and a McCleod gage. The entire system could be easily pumped down to a "stick" vacuum when the furnace tube was at room temperature; however, when the furnace tube was at 1120°C, the ultimate vacuum obtainable, as indicated by the McCleod gage, was of the order of 10^{-5} mm of Hg.

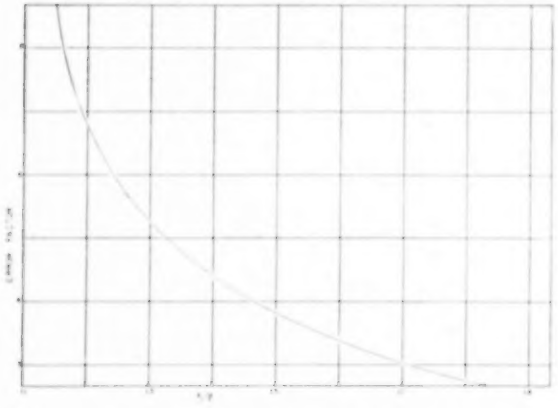

Fig. 2—Error factor as a function of the drop dimensions, X/Z.

The Optical System: The incandescent sessile drop was viewed through an optically flat window (surfaces parallel to 30 sec of arc) cemented at the end of the furnace-tube assembly with "Glyptal" cement. For the production of a well-corrected image, it is essential that such a window have optically flat surfaces of good parallelism. The camera, which was mounted directly behind the furnace-tube assembly, consisted of an F:9 "Artar" apochromatic photoengraving lens of focal length 14 in., and a plate holder mounted at opposite ends of a brass tube. Focusing was accomplished by means of a screw-thread adjustment which permitted the entire camera assembly to be moved parallel to the optical axis. The plate distance was fixed at about 42 in., giving an object distance of approximately 21 in., and a magnification of about 2.

Due to the sensitivity of the highly fluid molten drops to vibration, it was found necessary to place the bench, on which the apparatus was mounted, upon sponge-rubber mats.

Preparation and Purification of Materials: All of the copper samples used in these measurements were cut from a single piece of highly purified electrolytic copper, supplied by the Research Dept. of Phelps Dodge Corp. The reported analysis indicated that all impurities other than oxygen totaled 0.0053 pct. Each sample was first cleaned in 1:1 HNO_3, then in 1:1 HCl, and finally rinsed in distilled water. Copper-plated tools were used in all subsequent handling of the sample. The Cu_2S was prepared by first precipitating CuS from dilute sulphuric acid solution of reagent grade $CuSO_4 \cdot 5H_2O$ with H_2S. The filtered and washed precipitate was then mixed with sulphur and ignited in an alumina crucible at 1200°C for 30 min. Analysis of the resulting matte indicated 21.7 pct S.

Experimental Procedure: The weighed sample strip (1 to 3 g) was wound into a tight spiral and placed on the plaque, a 1x1 in. clear-silica plate polished to a mirror finish. A weighed amount of Cu_2S was deposited within the center of the spiral, the plaque and sample were placed on the silica plaque support and moved to the center of the furnace tube. The apparatus was assembled, opened to the vacuum pumps, and the heating current was turned on. During the heating period, the plaque surface was carefully leveled by rotating the furnace tube about its axis until tests of the image on the ground glass, using a small level, indicated the surface of the plaque to be horizontal in the direction at right angles to the optical axis. Leveling of the plaque along the optical axis could be accomplished by

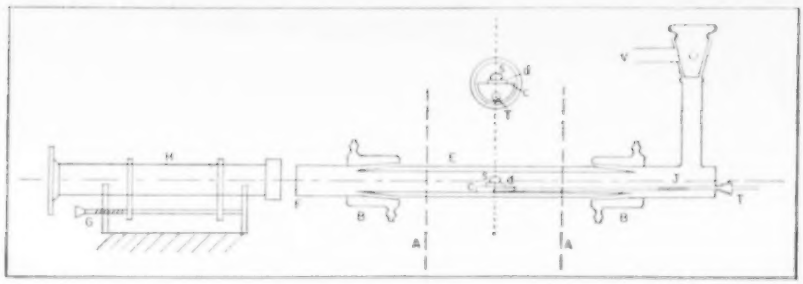

Fig. 3—Sketch of the apparatus for sessile-drop measurements. (Not to scale)

A—Section enclosed by furnace
B—Water-cooled pyrex joints
C—Plaque support (silica)
D—Silica plaque
E—Silica furnace-tube
F—Optical window
G—Camera focusing screw
H—Camera
T—Thermocouple well

slightly raising or lowering the free end of the furnace-tube assembly, and then focusing on the ground glass the front and back edges of the plaque to check their relative heights. The system was pumped out continuously until the temperature reached 700° to 800°C and then was flushed with several batches of dry hydrogen. The hydrogen pressure was finally adjusted to 50 mm Hg. The heating rate was decreased when the melting point was reached since it was found that movement of the sample, caused by rapid and uneven melting, often resulted in the sample falling off the plaque. The drop was maintained at 1120°C for 15 min before the first photograph was taken. The image of the drop was very carefully focused on the ground glass with the aid of a 15-X microscope eyepiece placed on the clear central portion of the glass. Focusing at full aperture could be done with an accuracy of at least ±¾ turn of the focusing screw (±0.64 mm) which permits a reproducible magnification to within ±1.2 parts per thousand. Usually two photographs were taken at a 15-min interval. Immediately after the last exposure, the heating current was turned off and the system cooled to room temperature. Kodak "M" plates (3¼x4¼ in.) were used with an exposure of ½ sec at f:16, and were developed in Kodak "Microdol."

Measurements and Calculations: The microcomparater (a "Toolmaker's Microscope," manufactured by Bausch and Lomb) used in this work was provided with micrometer screw adjustments for measuring both horizontal and vertical distances on the plates. The plate to be measured was mounted on the instrument with the base of the drop-image parallel to the horizontal axis of the stage. The only measurement which presented any difficulty was that of Z (Fig. 1). Because of the uncertainty in locating the vertical coordinate of the tangent point, it is by far the less accurate of the two measurements required for the surface tension calculation. The procedure which was used to locate the point b is as follows (cf. Fig. 4): The horizontal micrometer screw was adjusted until the vertical cross hair was coincident with the edge of the drop in the region of the tangent point. The plate was then moved horizontally a distance c (e.g., 0.005 in.), such that the vertical cross hair intersected the drop surface above and below the tangent point. The vertical coordinates (d and e) of these intercepts were measured; then the stage was again moved horizontally the fixed distance c in the same direction and the new intercepts read. The process was repeated until five sets of coordinates were obtained for each side of the drop. The coordinate f in Fig. 4 is the mean value of the coordinates d and e. Corresponding f values for each edge of the drop were averaged and plotted against the distance the vertical cross hair had been displaced from the edge of the drop. The resulting nearly straight line was extrapolated to zero to obtain b. The reproducibility of the Z values obtained in this way was about 4 to 6 ppt (parts per thousand), while the X values could be measured to at least 1 ppt. The accuracy of the $2a$ and h values (Fig. 1) was limited to approximately 5 ppt by the relatively poor definition of the image in the region of the base of the drop. The calculation of the data, while straightforward, is too lengthy to be presented here. The method is described by Ellefson and Taylor.[3] A liquid-density value of 7.90 g per cu cm was used in all the calculations.[4] While the presence of 0 to 0.8 pct S in liquid copper will quite probably change the density, it is believed that this effect will be quite small. Attempts were made to determine the density of the individual liquid drops by calculation of the volume and measurement of the weight of the solidified drops, but the results were scattered and generally low. The lack of precision, while caused in part by the poor accuracy of the a and h values, resulted mainly from gas bubbles trapped within the drop at the liquid-solid interface (vida infra).

Experimental Results: Surface tension measurements of liquid copper drops containing 0.0055 to 0.83 pct S, as well as measurements on pure copper drops, are presented in Table I. The surface tension lowering, F, is plotted as a function of final sulphur concentration in Fig. 5.

Preliminary sessile-drop measurements, made by means of a less refined apparatus, served to show that the surface tension of copper in vacuo is not

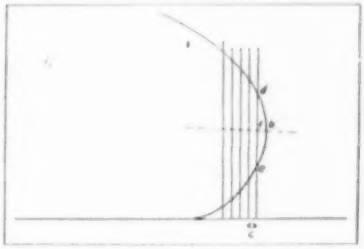

Fig. 4—Sketch showing method of locating the vertical coordinate of the tangent point on a liquid drop.

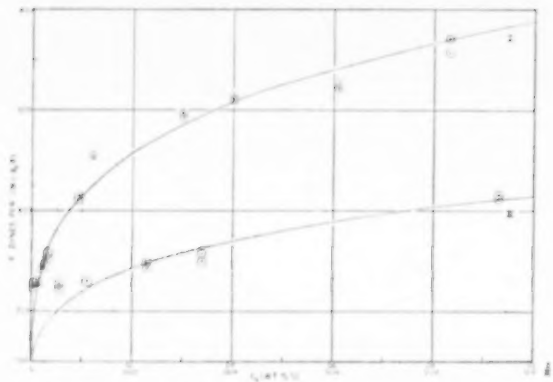

Fig. 5—Surface-tension lowering, F, of liquid copper as a function of the sulphur content. Circles are experimental points; the solid line is calculated from the relation: $\log C_s = 3.9 \log F - 11.27$.

greatly different from the values shown in the table for the sulphur-free samples. Measurements in the presence of nitrogen and argon were inconclusive because of traces of oxygen in the system, partly as impurities in the gases and partly from the permeability of the silica tube to air at the operating temperature. The presence of a trace of oxygen in the system led to irreversible slagging at the copper-silica interface and a dull appearance of the cooled drop. This slagging was a remarkable effect in that the cooled drop adhered to the plaque so strongly that when it was pried loose a quantity of silica adhered to the base of the drop. On the basis of the less accurate measurements, it appears that in cases of appreciable slagging, the surface tension of copper is lowered about 100 to 300 dynes per cm. Only when the sample was melted in hydrogen was slag formation eliminated. While the cooled drop often adhered to the plaque, it usually could be removed easily, leaving the silica surface unaffected, and the copper surface smooth and shiny. The variability of the contact angles listed in the table may be caused in part by traces of slag formation, but the agreement among the three runs on pure copper indicates the surface tension effect to be small.

With one exception, chemical analyses* of the cooled drops indicate that an appreciable loss of sulphur occurred during the course of each run. Rough calculations, based on the Cu_2S volatility data of Hsiao and Schlechten,[5] indicate that losses due to volatilization and decomposition of Cu_2S during the heating period, prior to the introduction of hydrogen were not appreciable.

* These microanalyses were performed by Dr. C. Tiedcke, Teaneck, N. J. An accuracy of ± 2 pct of the absolute value is claimed.

While considerable losses of sulphur to the gas phase may be expected as a result of the reaction

$$Cu_2S + H_2 = 2Cu + H_2S \qquad [1]$$

calculations based on the data of Sudo[6] and Rosenqvist[7] for the equilibrium of the reaction

$$S\ (\text{dissolved in liquid Cu}) + H_2 = H_2S \qquad [2]$$

indicate that after the sample has melted the total amount of H_2S in the gas phase would, in general, decrease to less than 2 pct of the sulphur content of the drop. However, inspection of the furnace tube after each run revealed small deposits of copper and copper sulphide which had condensed on the cooler portions of the inner wall. In addition, appreciable amounts of material were thrown off from the sample during the melting process, even when the heating rate was reduced to a practical minimum. It is probably this effect, along with transfer of sulphur to the cooler portions of the furnace tube, which is primarily responsible for the generally lower final sulphur contents, especially since the undissolved Cu_2S in a partially melted sample floats on the molten surface as small crystals which are easily thrown off by surface eruptions. To decrease transfer of sulphur to and from the walls, the tube was frequently cleaned (with only partial success) and generally the runs were made in order of increasing sulphur content. An exception to this procedure was run No. 4. While this run was preceded by careful cleaning of the tube, it is apparent that sulphur was transferred to the drop during the run. The final sulphur content of the samples has in every case been taken as that corresponding to the measured surface tension values, since the good agreement between the duplicate surface tension measurements for each run indicates that no appreciable change in sulphur content occurred after the sample had melted, and since little change in composition could occur after the cooling drop had solidified.

The dimensions of a given drop often changed considerably between successive photographs in such a way as to suggest that a gas bubble was trapped in the drop. Indeed upon examination of the solidified drop, large cavities were often found at the metal-silica interface. In view of the large contact angles found for these drops, it is not surprising that such gas bubbles are stable. However, their presence would not be expected to affect the measured surface tension of these drops and, indeed, this is borne out by the general agreement between the values found for a given sample at successive time intervals.

It is clear from Fig. 5 that the presence of sulphur as an impurity in the pure copper samples would affect the surface tension values, and make necessary an additive correction to all of the F values. However, such a correction would not affect the

Table I. Surface Tension of Liquid Copper

Temperature = 1120° ± 3°C Atmosphere = 50 mm (Hg) of H_2

Run No.	Sulphur, Pct Start	Sulphur, Pct Final	Surface Tension, Dynes per Cm	Surface Tension Lowering, Dynes per Cm, F	Calculated Contact Angle, Degrees
1	—	—	1250±20		132
2	—	—	1288±20		148
	—	—	1290±20	1269±20	131
3	—	—	1264±20	0	152
			1250±20		
4	0.00065	0.0055	1025±13	244	163
			1016±13	253	
5	0.0126	0.011	1012±15	257	172
			1009±15	260	172
6	0.0225	0.023	983±13	286	164
			971±13	298	171
7	0.0429	0.034	968±14	301	158
			953±14	316	
8	0.122	0.093	839±11	430	155
			847±11	422	156
9	0.227	0.12	758± 9	511	167
			759± 9	510	167
10	0.375	0.30	676±11	593	155
			683±11	586	154
11	0.564	0.40	639± 9	630	169
			651± 9	618	166
12	0.754	0.61	613± 9	656	165
			627±10	642	164
13	1.06	0.83	529± 7	740	154
			555± 7	714	156

quantity dF/dc, which appears in the Gibbs equation. Impurities other than sulphur in the pure copper sample, will likewise have a negligible effect on dF/dc, since they are present in virtually the same concentration in all of the samples.

The uncertainties indicated in the table for the surface tension values have been obtained by use of the curve in Fig. 2. The corresponding uncertainties in the surface tension lowering, (F), are represented approximately by the size of the circles in Figs. 5 and 6.

Discussion of Results

The value listed in Table I for the surface tension of pure copper in hydrogen at 1120°C (1269 dynes per cm) is considerably higher than previously reported values. Thus, Sauerwald, et al.[5] report 1150 dynes per cm at 1115°C; Becker, Harder and Kornfeld,[2] who used the sessile-drop method with the sample in argon, find 1038 dynes per cm at 1150°C. Since the effects of various gases such as nitrogen, hydrogen, and argon on liquid-metal surface tensions are by no means well known, it is not possible at this time to compare critically these different values for copper. However, when conflicting values of surface tension of a pure liquid are encountered, the highest value should be preferred, since traces of impurities may well lower the surface tension appreciably, but can have only a negligible effect toward raising it.

Application of the Gibbs equation to the system Cu-S-H at constant temperature leads to the result

$$-d\gamma = dF = \Gamma_1\, d\mu_1 + \Gamma_2\, d\mu_2 + \Gamma_3\, d\mu_3 \qquad [3]$$

wherein Γ_1, Γ_2, and Γ_3 denote the excess surface concentrations of copper, sulphur, and hydrogen, and μ_1, μ_2, and μ_3 denote the partial molal free energies of the three components. For the dilute solutions of sulphur used in the present experiments it may be assumed that the activity of copper is constant. The results of Schuhmann and Moles[6] establish that even in a saturated solution of sulphur in liquid copper at these temperatures, the H_2S/H_2 ratio in equilibrium with the solution is approximately 4×10^{-8}. Since the present experiments were conducted at a constant H_2 pressure (50 mm Hg), the hydrogen activity in the system may reasonably be assumed constant. Thus, Eq. 3 reduces to

$$dF = \Gamma_2\, d\mu_2 \qquad [4]$$

By incorporating the approximate proportionality of the sulphur activity to the sulphur concentration, reported by Rosenqvist[7] and Sudo,[8] Eq. 4 becomes

$$dF = \Gamma_2\, RT\, d \ln C_2 \qquad [5]$$

where C_2 denotes the pct S in the liquid copper. This is the so-called approximate form of the Gibbs equation. Two rearrangements of Eq. 5 will aid in the correlation of the data:

$$\frac{dF}{dC_2} = \frac{\Gamma_2 RT}{C_2} \qquad [6]$$

$$\frac{d \ln C_2}{d \ln F} = \frac{F}{\Gamma_2 RT} \qquad [7]$$

In low temperature systems it is often the case that F is proportional to C_2 at low concentrations. Thus in Eq. 6 dF/dC_2 may be replaced by F/C_2 and the following relation is obtained:

$$\frac{F}{\Gamma_2} = RT, \text{ or } FA_2 = RT \qquad [8]$$

where A_2 is the reciprocal of Γ_2, or the area per mol of adsorbed material. Systems which behave according to Eq. 8 have been interpreted on the basis of a surface film of the order of one molecular diameter in thickness in which the adsorbed substance behaves as an ideal two-dimensional gas.

It is evident from Fig. 5, where F is plotted as a function of C_2, that even in the very low concentration region, the relation of F to C_2 is far from linear. The ideal two-dimensional gas model, represented by Eq. 8, is, therefore, not applicable to these data.

In Fig. 6, the data are plotted as log C_2 vs log F. Within the uncertainty of the data, the relation is

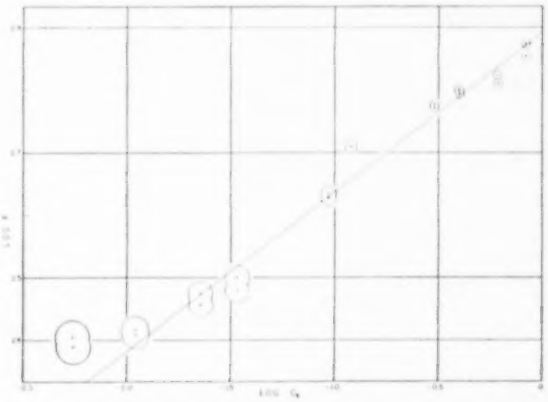

Fig. 6—Log F vs log C_2. Circles are experimental points. The straight line is expressed by: log C_2 = 3.9 log F — 11.27.

linear with a slope of 3.9. The deviation shown by the lowest point is not serious in view of its uncertainty. The deviation of the point at 0.12 pct S might well be disregarded on the grounds that the sulphur analysis for this sample is anomalous; thus, it is much lower relative to the starting composition for this run than are any of the other analyses. The curves in Fig. 5 have been constructed from the straight-line plot in Fig. 6 ($\log C_2 = 3.9 \log F - 11.27$) and the fit is quite satisfactory except for the two points already mentioned.

If the experimentally found value of $d \ln C_2/d \ln F$ is now incorporated in Eq. 7, the following relation is obtained:

$$3.9RT = \frac{F}{\Gamma_2}$$

or

$$3.9RT = FA_2 \qquad [9]$$

Eq. 9, which correlates the present data, bears a formal relationship to the Schofield and Rideal equation[10]

$$F\,(A_2 - B) = iRT \qquad [10]$$

which purports to describe the behavior of a non-ideal gaseous adsorbed-film on a liquid surface. In Eq. 10 B is a constant which represents the limiting area of the surface-active material, and $1/i$ is a measure of the lateral molecular-cohesion. Eq. 9 reduces to Eq. 10 if B is zero and i is 3.9. A value of i greater than unity is meaningless, however, since it would indicate a molecular repulsion rather than a cohesion, hence the Schofield-Rideal equation does not aid in the interpretation of the present results.

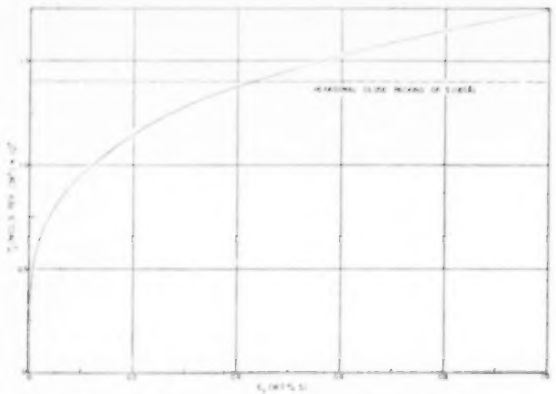

Fig. 7—The surface-excess concentration of sulphur, Γ_s, as a function of the bulk concentration of sulphur, C_s. Calculated from the relation: $\Gamma_s = KC_s^{1/3.9}$. The horizontal dashed line represents the surface-excess concentration of sulphur which would result from a close-packed monolayer of sulphur atoms.

The relation between the bulk concentration, C_s, and the surface excess-concentration, Γ_s, which applies to the present data may be derived as follows:

From Eq. 9

$$\log \Gamma_s + \log 3.9RT = \log F \qquad [11]$$

If Eq. 11 is combined with the experimental relation

$$\log C_s = 3.9 \log F - 11.27$$

the result is

$$\log \Gamma_s = \frac{1}{3.9} \log C_s - \log 3.9RT + \frac{11.27}{3.9}$$

or

$$\Gamma_s = KC_s^{1/3.9} \qquad [12]$$

where $K = \frac{782}{3.9RT}$ and has the value 1.74×10^{-9} when R is expressed in ergs per mol per °K. Fig. 7, which shows graphically the relation between C_s and Γ_s, was calculated from Eq. 12.

Based on the recent measurements of Rosenqvist[7] and Schuhmann and Moles[9] the solubility of sulphur in liquid copper is probably about 1 pct at 1200°C. If this value is used without correction for the present work at 1120°C, then Fig. 7 indicates the maximum surface excess of sulphur to be 1.74×10^{-9} mols per sq cm. This compares favorably with the value expected for a hexagonal close-packed monolayer of sulphur atoms — 1.4×10^{-9} mols per sq cm (calculated with a van der Waals radius of 1.85Å), represented by the horizontal dashed line in Fig. 7. In all probability the surface excess of sulphur is not confined to a monolayer, hence the fact that the calculated surface excess exceeds the amount that can be held in a close-packed monolayer does not invalidate the results.

The adsorption equation (Eq. 12) found to apply to the present data is identical to the Freundlich adsorption isotherm, and is similar in form to the equation which represents the distribution equilibrium of a solute between two phases when the solute is associated in one phase into groups of n atoms (n is 3.9 in Eq. 12). This suggests one possible model which would correlate the results in the Cu-S adsorption system. If the sulphur in the bulk solution is associated into groups containing four atoms of sulphur, while the sulphur in the surface film is unassociated, then the distribution between the bulk and the surface would take the form

$$k = \frac{\text{activity of sulphur in bulk}}{(\text{activity of sulphur in surface})^4}$$

Assuming a Henry law relationship to apply,

$$k' = \frac{C_s}{\Gamma_s^4}, \text{ or } \Gamma_s = k'' C_s^{1/4} \qquad [13]$$

Association of sulphur into groups of four atoms is unlikely, however, in view of the results of Sudo[6] and Rosenqvist[7] on sulphur activity.

A further correlation that may be significant is the near equality of the free-surface energy of adsorption per atom of sulphur, $\frac{3.9RT}{N_{Avg.}}$, which is 7.5×10^{-13} ergs, and the free surface energy of pure liquid copper per atom of copper, which is 7.8×10^{-13} ergs.† The agreement between these values suggests that, atom for atom, the adsorbed sulphur dissipates nearly all of the free electrovalent forces of copper which give rise to the surface tension.

† The number of copper atoms per sq cm of surface = $0.918 \cdot \left[\frac{N_{Avg.} \cdot d}{M}\right]^{2/3} = 1.63 \times 10^{15}$, where d is the liquid density and M the atomic weight of copper. The factor 0.918 is a packing factor for hexagonal closest packing.

$$\frac{\text{Surface tension of pure copper}}{\text{No. of copper atoms per sq cm}} = \frac{1269}{1.63 \times 10^{15}} = 7.8 \times 10^{-13} \text{ ergs.}$$

More experimental data on this and similar systems are needed in order to establish a consistent model for the surface-active behavior of the group VI elements in liquid copper. Preliminary experiments with selenium as an impurity[11] have shown this element to behave in a manner almost identical to sulphur. Experiments with oxygen as the impurity have failed due to inability to find a solid surface which is inert towards liquid copper containing dissolved oxygen.

Acknowledgment

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Metallic Oxidation in Chromium Steel Melting

by D. C. Hilty, G. W. Healy, and Walter Crafts

By means of a theoretical extension of the Cr-C temperature relation in molten chromium steels to low chromium contents and by a correlation of the ratios of chromium to iron in the slag and metal, a method has been developed for estimating the amount of metallic oxidation during the oxidizing period of a chromium steel heat. Application of this method has indicated that due to temperature limitations metallic oxidation may be excessive for very low carbon steels charged with more than a small amount of chromium.

IN the melting and refining of chromium steels the oxidation of carbon to a low level is accompanied by the oxidation of chromium and iron in considerable amounts. For the subsequent recovery of chromium and associated iron from the slag, the amounts of metal oxidized must be known for efficient reduction and control of composition. Further, in order to arrive at an accurate understanding of the process, so that chromium-bearing scrap can be used effectively, information is required on the relation between composition of the charge and weight of metal oxidized to reach the desired carbon content.

Crafts and Rassbach[1] recently developed an empirical relation between chromium, iron, and manganese oxidized per ton of steel charged, and final carbon content and temperature. This relation did not show any dependence on amount of chromium charged. However, it was felt that such a dependence might be present, so a further analysis of the data was undertaken.

Cr-C Relation at Low Chromium Levels

Crafts and Rassbach's results were based on temperatures estimated from the Cr-C relation previously established experimentally by Hilty,[2] and on a single temperature observation (immersion thermocouple) on a heat made at low temperature from a charge of carbon steel scrap containing only a small amount of residual chromium. The temperatures estimated from the Cr-C relation were for heats charged with chromium-bearing scrap and which contained substantial amounts of chromium (2 to 10 pct) at the end of the oxidizing period. No data, other than that from the low temperature heat just mentioned, were available for the low and intermediate chromium ranges.

In order to extend the range in which temperature may be estimated, the Cr-C relation requires further consideration. The Cr-C relation, expressed as

$$K = \frac{\%\ \mathrm{Cr}}{\%\ \mathrm{C}};\ \log K = -\frac{15{,}200}{T} + 9.46 \qquad [1]$$

suggests that carbon is zero at zero chromium in the bath. This is obviously absurd, because as chromium approaches zero the carbon content must approach the limit established by the Fe-C-O equilibrium. It is evident, therefore, that the Cr-C relation as defined by Eq. 1 is not valid below some minimum chromium content that may vary with temperature. On the other hand, it is probable that any such limiting chromium content is less than 4 pct at 3200°F, since the experimental data from which the Cr-C relation was derived included several observations at that level with no deviation. In any event, it is apparent that further evaluation of the results presented by Crafts and Rassbach is dependent upon a means for estimating the Cr-C temperature relation in the low chromium region.

From the observations of Chen and Chipman[3] and thermodynamic data given in *Basic Open Hearth Steelmaking*,[4] the following relation can be calculated for melts containing small amounts of chromium and carbon assuming a carbon monoxide pressure of 1 atm:

$$K' = \frac{\%\ \mathrm{Cr}}{(\%\ \mathrm{C})^2};\ \log K' = \frac{-21{,}250}{T} + 13.88 \qquad [2]$$

Moreover, by plotting Eq. 2 for any given temperature on cartesian coordinates, it can be extrapolated graphically to the limiting carbon content at zero chromium for the specified temperature without serious complications.

By means of Eq. 2 and its extrapolation, the Cr-C temperature relations originally established experi-

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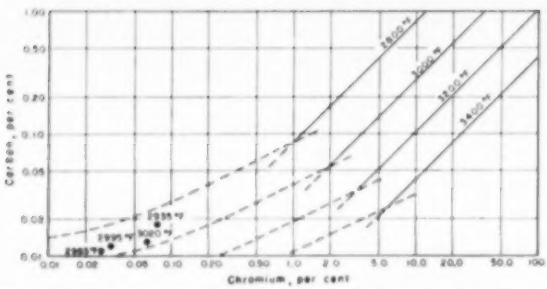

Fig. 1—Approximate Cr-C-temperature relations for all chromium contents of interest in the melting of chromium steels.

mentally at moderate to high chromium levels can be extended through the range of low chromium concentration as shown in Fig. 1. It is emphasized that the dashed curves of Fig. 1 are only approximations of the true Cr-C relations at low chromium contents. Nevertheless, they appear to be of a reasonable order and are considered quite useful for practical purposes.

Table I. Analytical Results on Samples

Temperature, °C	Carbon, Pct	Manganese, Pct	Chromium, Pct
2934	0.018	0.010	0.058
3020	0.013	0.006	0.048
2995	0.012	0.006	0.026
2985	0.011	0.006	0.021

Effect of Manganese

The similarity in the relative deoxidizing powers of manganese and chromium in liquid iron is demonstrated by the work of Hilty and Crafts[5] on manganese and that of Chen and Chipman on chromium. Moreover, Hilty pointed out the resemblance between the heat of formation calculated for the chromous oxide assumed in order to explain his experimental results and that of manganous oxide. In addition, arc-furnace heats made by oxygen-blowing practice from charges containing several percentages of manganese as well as chromium-bearing scrap have contained total chromium plus manganese contents at the end of the oxidizing period equal to the chromium contents alone in similar heats made in the same furnace from low manganese charges oxidized to the same carbon levels with the same rate of oxygen input. Consequently, it is considered that the moderate concentrations of manganese usually found in chromium-steel heats are substantially equivalent to similar quantities of chromium and should be added to the chromium for practical applications of Fig. 1. They have been so considered in the arc-furnace data described below.

In this regard it is notable that the four data points spotted in at the very low chromium level in Fig. 1 refer to a laboratory heat that contained a small, but presumably effective, amount of manganese. These data were obtained from a 100-lb induction furnace heat charged with Armco iron and a small amount of high carbon ferrochromium. The heat was held at relatively constant temperature in the open. Samples were taken by the Taylor sampling technique and temperatures were measured by means of Pt-Pt-10 pct Rh thermocouples. Analytical results on the samples are given in Table I. These results agree fairly well with the curves in Fig. 1. The agreement, however, may be fortuitous because of analytical uncertainty and limited understanding of the controlling reactions.

Effect of Metal Composition on Ratio of Cr + Mn to Fe in Slag

Fairly complete data for 27 arc-furnace heats described by Crafts and Rassbach were available. These heats were made in a 2000-lb capacity, magnesia-lined, arc furnace from charges containing various amounts of chromium and nickel as shown in Table II. After meltdown, all heats were blown with oxygen in order to lower their carbon contents. Analytical results from slag and metal samples taken at the end of the oxidizing period are tabulated in Table II.

Table II. Experimental One Ton Steel Heats

Heat	Charge Composition: Charge Weight, Lb	Cr Charged, Lb*	Mn, Pct in Charge†	Ni Charged, Lb	Metal Analysis after Oxygen Blow, Pct: Ni	Cr	Mn	C	Slag Analysis after Oxygen Blow, Pct: Cr	Mn	Fe	SiO_2	MgO	CaO	Al_2O_3	Lb Cr+Mn+Fe Oxidized per Ton
1	2099	242	0.50	208.0	10.86	6.16	0.21	0.018	29.25	5.54	12.22	18.76	10.28	8.86	—	190
2	2063	241	0.50	208.5	10.95	7.23	0.23	0.036	19.35	4.08	8.96	15.06	13.16	19.35	—	168
3	2109	241	0.50	207.0	10.13	9.39	0.54	0.065	11.37	5.05	4.45	30.20	17.41	22.54	—	86
4	2045	231	0.50	257.0	12.84	8.80	0.38	0.048	14.04	2.76	4.47	24.48	14.22	29.67	1.36	80
5	2075	230	0.50	257.0	13.07	7.92	0.26	0.048	15.10	1.52	6.14	19.38	26.54	19.74	1.50	108
6	2070	231	0.50	257.0	13.37	8.33	0.40	0.034	20.38	3.86	6.25	21.16	14.77	20.86	4.98	103
7	2070	232	0.50	260.0	13.12	8.65	0.40	0.044	12.42	2.74	4.42	24.38	22.43	21.70	3.53	92
8	2090	232	0.50	257.0	13.23	8.44	0.36	0.060	15.68	1.90	5.37	21.50	17.53	22.60	2.45	96
9	2014	164	0.50	257.0	14.17	3.34	0.14	0.016	22.80	3.40	19.50	13.58	10.44	13.29	0.96	206
10	2026	164	0.50	257.0	13.91	3.89	0.15	0.020	18.82	2.74	17.50	14.18	18.62	10.09	4.32	188
11	2014	164	0.50	257.0	14.21	2.80	0.08	0.015	24.42	3.19	22.72	15.50	7.54	9.14	—	236
12	2038	166	0.50	260.0	14.02	3.36	0.12	0.022	14.41	2.20	17.42	14.08	22.40	9.76	—	240
13	1795	111	0.50	235.5	15.06	1.59	0.05	0.016	18.19	2.78	28.60	14.72	3.95	13.12	1.26	262
14	1891	133	0.50	236.5	13.60	2.93	0.11	0.017	17.27	3.42	20.13	17.40	8.09	17.50	2.08	201
15	1760	123	0.50	—	—	1.98	0.05	0.020	17.44	1.55	26.35	16.36	14.36	6.62	1.66	272
16	1681	98	0.50	—	—	1.83	0.07	0.026	9.08	1.49	17.20	16.60	30.34	15.03	4.46	260
17	1660	98	0.50	—	—	2.28	0.06	0.022	13.22	1.36	21.50	16.94	20.18	12.66	2.10	212
18	1640	98	0.50	—	—	2.20	0.07	0.015	11.90	1.39	23.85	15.58	16.32	14.22	2.40	250
19	1576	9	0.50	—	—	0.03	0.02	0.018	2.82	1.43	37.0	15.00	16.84	12.83	1.80	230
20	1576	9	0.50	—	—	0.16	0.03	0.020	1.88	1.94	41.3	12.56	2.90	11.06	2.30	213
21	1576	9	0.50	—	—	0.16	0.04	0.013	1.50	1.81	38.8	15.06	17.61	8.28	2.04	239
22	1576	9	0.50	—	—	0.06	0.02	0.018	0.50	1.32	45.1	10.80	18.52	7.86	2.40	303
23	1576	9	0.50	—	—	0.08	0.02	0.018	0.89	1.32	43.5	11.80	16.30	8.64	4.60	269
24	1576	9	0.50	—	—	0.04	0.04	0.012	0.75	1.62	45.5	12.00	13.26	9.98	2.80	249
25	1576	9	0.50	—	—	0.14	0.02	0.014	2.10	1.03	34.2	12.40	21.81	10.43	6.80	307
26	1576	9	0.50	—	—	0.18	0.03	0.012	2.20	1.56	40.3	15.58	15.58	9.87	2.12	243
27	1322	4.1	0.34	35.8	3.47	0.02	0.01	0.017	0.52	0.59	47.4	14.70	13.19	5.27	5.84	525

* Charge weight plus 9 lb average pickup from lining except Heat 27, obtained from material balance.
† Estimated 0.50 pct Mn in charge except Heat 27, obtained from material balance.

Ratios of the chromium plus manganese to iron contents of the slag were plotted as a function of those of the metal with the result illustrated by Fig. 2.

The data are adequately represented by the line of regression whose equation is

$$\log \frac{(\%\,Cr + \%\,Mn)}{(Fe)} \text{ in slag} = 0.812 \log \frac{[\%\,Cr + \%\,Mn]}{[\%\,Fe]} \text{ in metal} + 1.238 \qquad [3]$$

Although it appears that the points might be fitted better by two intersecting lines, such a representation is not justified statistically.

The excellent correlation evident in Fig. 2 is rather surprising, especially since the heats covered rather broad ranges of both composition and, presumably, temperature. Apparently however, the relative proportions of chromium (plus manganese) and iron in the slag are controlled almost entirely by the amounts of chromium (plus manganese) and iron in the metal without regard to carbon content or apparent temperature.

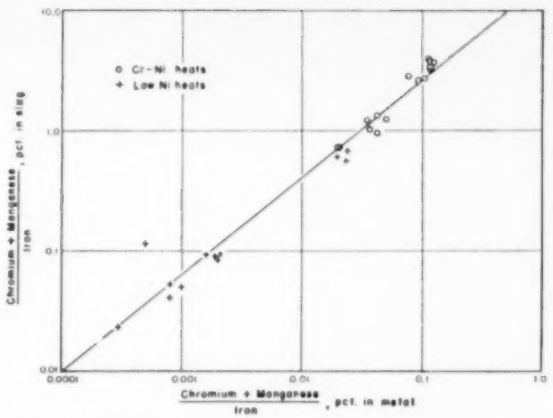

Fig. 2—Correlation of rates of chromium plus manganese to iron in slag and metal.

Weight of Metal Oxidized

The relation shown in Fig. 2 considerably simplifies the task of calculating the weight of metallics oxidized during the oxidizing period of a chromium-steel heat. In order to perform such a calculation, it is necessary to know only the amounts of chromium and manganese in the charge in addition to the metal composition at the end of the oxidizing period according to the following formula, the derivation of which is shown in the Appendix:

Pounds (Cr + Mn + Fe) oxidized per ton of charge

$$= 2000 \frac{(Cr_1 + Mn_1) - (Cr_2 + Mn_2)}{\frac{100S}{1+S} - (Cr_2 + Mn_2)} \qquad [4]$$

where Cr_1 is the percentage of chromium in the charge; Mn_1, the percentage of manganese in the charge; Cr_2, the percentage of chromium in the bath after oxidation; Mn_2, the percentage of manganese in the bath after oxidizing; and $S = \frac{(\%\,Cr + \%\,Mn)}{(\%\,Fe)}$ in slag after oxidation which can be calculated from the bath composition by Eq. 3 or read from Fig. 2.

In Fig. 3 is shown the correlation of metallic oxidation of the heats in Table II calculated by Eq. 4 with the observations of Crafts and Rassbach. In general, the correlation is good. The deviations are easily explained by the rather arbitrary assumption of 0.50 pct Mn in the charge for most of the heats and by the lack of adequate tracers such as nickel to assist in calculating material balances for most of the low chromium, low nickel charge heats.

With the additional help of Fig. 1, for estimation of residual chromium after the oxygen blow, the amount of metallic oxidation can now be calculated for charges of varying chromium contents oxidized to specified carbon and temperature levels. The results of such calculations are shown in Figs. 4 and 5 for 0.015 and 0.05 pct C, respectively.

The substantial effect of charged chromium as well as the effects of the carbon content and temperature attained on the oxidation losses is apparent.

These illustrations strongly imply that avoidance of excessive oxidation loss in making very low carbon steel from a high chromium charge may involve temperatures beyond the scope of usual steelmaking furnaces. Consequently, it would appear desirable to regulate the amount of chromium in a charge with due regard to the carbon content of the steel being made and the maximum safe operating temperature of the furnace.

In rationalizing the above results, it is presumed that the Cr-C temperature relation of Fig. 1 refers approximately to a state of equilibrium wherein the metal phase is saturated with an oxide of chromium as well as carbon monoxide. If, however, the chromium content of the bath after oxidation is less than that specified by the carbon content and temperature so that the metal is not saturated with a chromium oxide—as may well be the case in heats made at fairly high temperatures from low chromium charges—then Fig. 2 and Eq. 4 do not apply. Under such conditions, the chromium content of the bath may be controlled by other factors such as slag basicity, etc., in a manner analogous to the case of residual manganese in the open hearth. It would also appear that the relations shown in Figs. 1 and 2

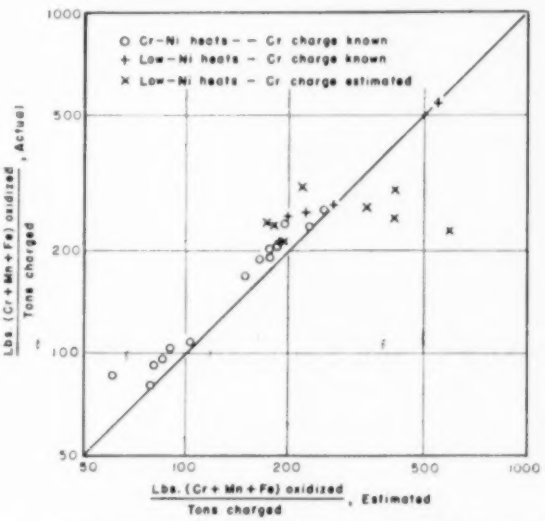
O Cr-Ni heats — Cr charge known
+ Low-Ni heats — Cr charge known
× Low-Ni heats — Cr charge estimated
Lbs. (Cr + Mn + Fe) oxidized / Tons charged, Actual
Lbs. (Cr + Mn + Fe) oxidized / Tons charged, Estimated

Fig. 3—Correlation of estimated with actual metallics oxidized.

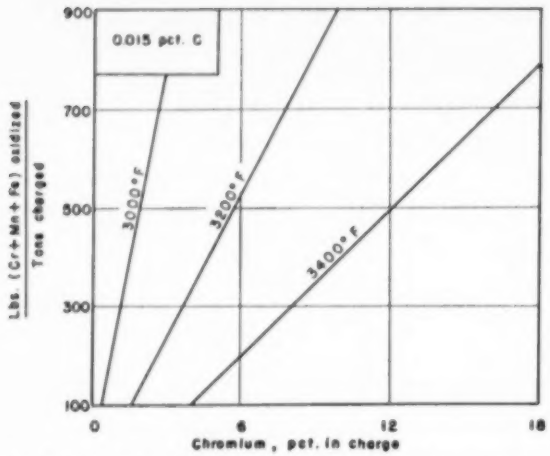

Fig. 4—Estimation of metal oxidized in blowing with oxygen to 0.015 pct C.

are of practical significance only when oxidation is carried out to such a degree that a significant proportion of the charged chromium is oxidized. For this reason the minimum amount of metal oxidized has been limited to 100 lb per ton in Figs. 4 and 5.

The temperature dependence of metallic oxidation indicated by Figs. 4 and 5 differs somewhat from the empirical relation reported by Crafts and Rassbach. Moreover, it is indicated to vary with the amount of chromium charged. A probable explanation of the differences is that among the heats used by Crafts and Rassbach to estimate the temperature coefficient most of the heats with higher chromium charges also reached higher temperatures at the end of the oxidizing period. Such a result would be expected for heats made under the same conditions with regard to oxygen input, etc., because of the relatively higher heat of oxidation of chromium. Thus, under practical operating conditions, the effect of increased chromium in the charge on metallic oxidation would be offset by the higher temperature attained during oxidation. The relation presented by Crafts and Rassbach is believed to illustrate the net result of regulating the amount of chromium in a charge with regard to the carbon content of the steel being made as suggested by Figs. 4 and 5 of this paper.

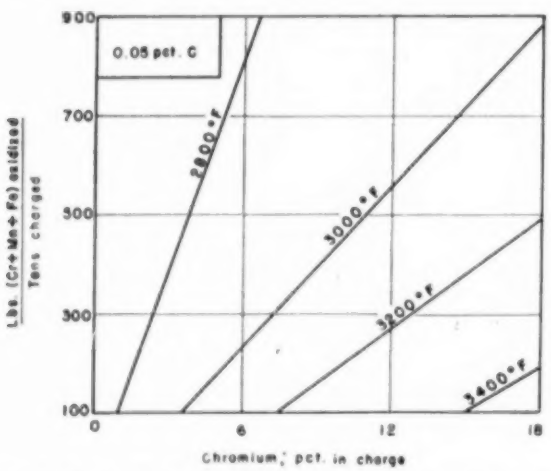

Fig. 5—Estimation of metal oxidized in blowing with oxygen to 0.05 pct C.

Summary

By means of published thermodynamic data an approximate Cr-C temperature relation for molten chromium steel under oxidizing conditions has been derived for the range of low chromium concentrations where the relation established experimentally at higher chromium levels is believed to be invalid.

The ratio of chromium (plus manganese) to iron in the slag of a chromium steel heat has been shown to depend almost entirely on the ratio of chromium (plus manganese) to iron in the metal. Through this relationship an expression for calculating the metallic oxidation during the oxidizing period of a heat has been derived.

It has been demonstrated that the amount of metallic oxidation is dependent upon the amount of chromium in the charge and the carbon content and temperature of the bath at the end of the oxidizing period. It is inferred that for steels oxidized to very low carbon contents from charges containing more than a small amount of chromium, metallic oxidation may be excessive because of maximum temperature limitations imposed by operating conditions.

Acknowledgments

The authors wish to express their appreciation for the valuable suggestions and criticism of John Chipman, Head, Dept. of Metallurgy, Massachusetts Institute of Technology, for the assistance received from H. J. O'Connor, Electro Metallurgical Co., in statistical studies, and particularly for the stimulating appraisal of the practical aspects of the problem by H. P. Rassbach, Electro Metallurgical Co.

Appendix

Derivation of Formula for Pounds of Metal Oxidized

Assume a charge weight of 2000 lb. Let the percentage of chromium plus the percentage of manganese in the charge be Cr_1. Then the pounds of chromium plus manganese in the charge are

$$\text{Lb Cr + Mn charged} = 2000 \times \frac{Cr_1}{100} = 20\,Cr_1 \quad [5]$$

Let X equal the pounds of chromium and manganese oxidized, and Y the pounds of iron oxidized. Then the weight of metal after oxidation will be

$$\text{Lb metal after oxidation} = 2000 - X - Y \quad [6]$$

It was shown that the ratio of chromium plus manganese to iron in the slag after oxidation can be estimated from the ratio of chromium plus manganese to iron in the metal (Fig. 2). Let S be the ratio of chromium plus manganese to iron in the slag. Then

$$\frac{\text{Lb Cr + Mn in slag}}{\text{Lb Fe in slag}} = \frac{X}{Y} = S \quad [7]$$

Eq. 6 can then be written in the form

Lb metal after oxidation =

$$2000 - X - \frac{X}{S} = 2000 - X\,\frac{1+S}{S} \quad [8]$$

Let Cr_2 be the percentage of chromium plus manganese in the metal after oxidation. The weight can then be calculated from Eq. 8.

Lb Cr + Mn in metal after oxidation =

$$\left(2000 - X\,\frac{1+S}{S}\right)\frac{Cr_2}{100} \quad [9]$$

But the amount of chromium plus manganese in the bath is also equal to the difference between the amount

originally charged (Eq. 5) and the amount oxidized designated by X.

$$\text{Lb Cr + Mn in metal after oxidation} = 20\,\text{Cr}_1 - X \quad [10]$$

Combining Eqs. 9 and 10, it is possible to solve for X

$$20\,\text{Cr}_1 - X = \left(2000 - X\frac{1+S}{S}\right)\frac{\text{Cr}_2}{100}$$

$$= 20\,\text{Cr}_2 - X\frac{1+S}{S}\frac{\text{Cr}_2}{100} \quad [11]$$

Hence

$$X\left(1 - \frac{1+S}{S}\frac{\text{Cr}_2}{100}\right) = 20\,(\text{Cr}_1 - \text{Cr}_2)$$

$$\text{Lb Cr + Mn oxidized: } X = \frac{(\text{Cr}_1 - \text{Cr}_2)}{1 = \frac{1:S}{S} - \frac{\text{Cr}_2}{100}}$$

or

$$X = 2000\frac{\text{Cr}_1 - \text{Cr}_2}{100 - \frac{1+S}{S}\text{Cr}_2} \quad [12]$$

Since total weight oxidized is $X + Y$ and since $S = X/Y$, the total weight oxidized becomes

$$X\left(1 + \frac{1}{S}\right) = X\frac{1+S}{S} \quad [13]$$

Multiplying Eq. 12 by this factor $\frac{1+S}{S}$ then results in

$$\text{Lb metal oxidized} = 2000\frac{\text{Cr}_1 - \text{Cr}_2}{\frac{100S}{1+S} - \text{Cr}_2} \quad [14]$$

In this equation Cr_1 is known from charge composition, Cr_2 from the analysis after oxidation, and S can be estimated from [% Cr + % Mn]/[% Fe] in the metal after oxidation, using Fig. 2.

Example: A heat containing 11.5 pct Cr and 0.5 pct Mn in the charge is blown with oxygen to 0.05 pct C, reaching a temperature of 3200°F. From Fig. 1 Cr_2 = 4.7 pct is obtained. If there is no nickel in the heat,

$$\frac{\%\,\text{Cr} + \%\,\text{Mn}}{\%\,\text{Fe}} = \frac{4.7}{100 - 4.7} = 0.049$$

Entering Fig. 2 at 0.049, S is estimated as 1.47. Then weight oxidized is

$$W = 2000\frac{(11.5 + 0.5) - 4.7}{\frac{100 \times 1.47}{1 + 1.47} - 4.7} = 266\text{ lb per ton charged.}$$

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Technical Note

Note on Contamination of Silicon Ingots

by G. Sandoz and H. E. Stauss

THE purpose of this note is to draw attention to the possibility that a melt may be contaminated by a material not in direct contact with it by means of gaseous intermediate agents. In recent years silicon has assumed a position of importance because of its use in rectifiers and transistors and close attention has been required to control of both purification and contamination during all stages of preparation.[1] This note reports the observation that a graphite heater may have contaminated silicon ingots without direct physical contact through reactions that developed gaseous products.

Several ingots made with the equipment of Hino and Stauss,[2] using a graphite heater in a dried helium atmosphere, were found to have a small amount of a very hard substance on the free surface. X-ray analysis by F. W. von Batchelder showed the substance to be SiC. Similar contamination was not found when the heater was tantalum, indicating that the carbon had not come from an original impurity in the silicon powder.

One source of the carbon could have been volatile matter released from the graphite heater. This heater had been used about ten times at the time of the observations. A second source of carbon could have been a direct reaction between the quartz crucible and the graphite heater to form CO, or an indirect reaction involving a crucible-melt interaction and volatile SiO as an intermediate product. Such reactions should have developed SiC on the heater; and in fact a yellow powder did form on the inside of the graphite heater although its composition could not be determined.

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TN 155E. Manuscript, Jan. 30, 1953; revision, March 6, 1953.

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Technical Note

Notes on the Plastic Critical Temperature In Strain-Induced Martensite Reactions

by C. Dean Starr

IT is well known that the martensite reaction can be induced to occur isothermally above the M_s temperature by plastic deformation of an alloy in its austenitic state, and also that as the test temperature is increased above the M_s temperature, plastic deformation under a given set of conditions merely causes additional slip with a corresponding tendency toward the formation of less martensite per unit of strain. Thus it appears that a maximum temperature exists above which martensite cannot be formed isothermally by plastic deformation. This temperature is commonly designated as the equilibrium temperature T_e.

Attempts to determine experimentally the equilibrium temperature have indicated that a critical temperature is observed, under a given set of conditions, above which martensite cannot be formed isothermally by plastic deformation. But the observation of a "so called" critical temperature, T_p, should not be confused with the equilibrium temperature, T_e.

In order to clarify this anomaly, true stress-strain curves were obtained for a series of type 304 stainless steel specimens (manufacturer's analysis, 0.08 pct C, 18 to 20 pct Cr, 8 to 10 pct Ni, balance Fe) over the temperature range 67° to 300°K.[1] Since martensite is magnetic, a quantitative measure of the amount of martensite that was produced by plastic deformation was obtained by determining the magnetic permeability as a function of strain. The results of these tests are given in Fig. 1. They appear to indicate that a plastic critical point does exist at about 240°K for the stainless steel under investigation here. But that 240°K is not the equilibrium temperature nor even a true plastic critical temperature can be seen from Fig. 2 wherein the permeability of the same material is shown to increase with cold rolling at 293°K.

Consequently the observation of a "so called" plastic critical temperature is dependent on the mode of deformation and has no theoretical significance.

Acknowledgment

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TN 153E. Manuscript, Nov. 21, 1952.

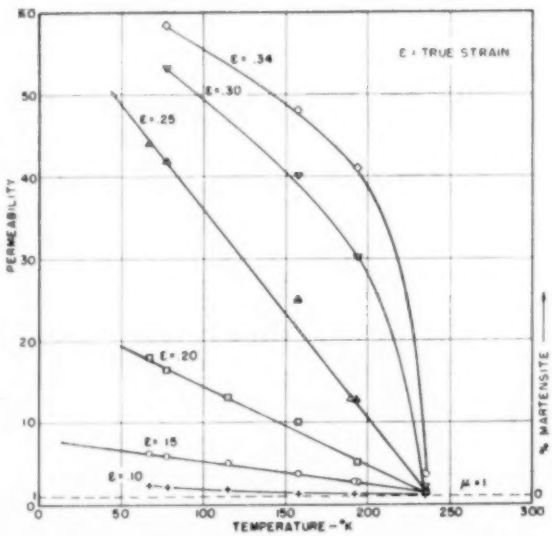

Fig. 1—An apparent plastic critical point as determined by simple tension testing.

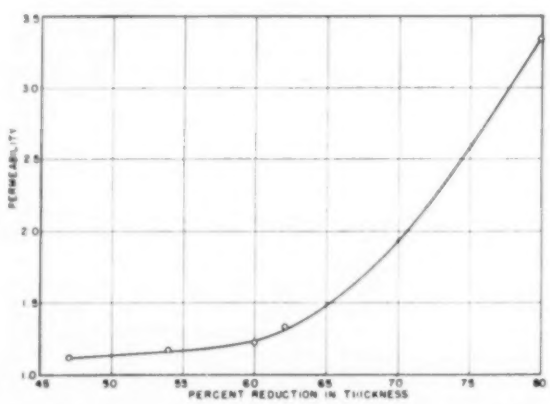

Fig. 2—Formation of martensite during cold rolling of 18-8 stainless steel at 293°K.

[1] C. Dean Starr, M. Reporter, J. E. Dorn: Investigations on the Mechanism of the Strain Induced Martensitic Reaction in 18-8 Stainless Steel. Univ. Calif. Inst. of Engineering Report, Series No. 22, Issue No. 15, June 1951.

Free Energy of Vaporization of Metals from 0° to 2000°C

by J. W. Evans

ONE of the most important and frequent calculations that the extractive metallurgist is called upon to make is that of the standard free energy change of a reaction (ΔF°). For many reactions of metallurgical interest, the lengthy arithmetical calculations have been eliminated by the use of free energy-temperature diagrams which have been constructed by Ellingham,[1] Richardson and Jeffes,[2] Kellogg,[3] and Osborn[4] (e.g., for oxides, sulphides, halides) from critical surveys of existing data. The speed and simplicity of making calculations with these graphs make them of great practical value and, this should be stressed, in the majority of cases no greater accuracy is obtained by the individual calculations from specific heat data, entropies, and heats of reaction.

With the increasing industrial application of vacuum methods to the winning of metals, notably at present magnesium and the alkaline earth metals, it is necessary to calculate low metal vapor pressures above reaction mixtures. The existing free energy diagrams usually give values where the solid and liquid metals are at unit activity. To obtain the free energy change associated with the reaction

$$\text{Metal (solid or liquid)} = \text{Metal (gas)}$$

it is customary to calculate this from either vapor pressure equations or from the appropriate free energy functions. This paper presents graphically the free energies of vaporization of a number of metals of economic importance which can be used in conjunction with other free energy diagrams. Certain metals have been excluded on the following grounds: titanium, tungsten, molybdenum, and platinum have very high heats of vaporization and boiling points and the uncertainty of these values arising from difficulties of high temperature measurements makes free energy values of doubtful accuracy; the data for cobalt, vanadium, and zirconium are uncertain; finally, arsenic, antimony, and bismuth could not be conveniently included because of the presence of mixtures of polyatomic and monatomic vapors in the temperature range considered.

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Discussion on this paper, TP 3476D, may be sent, 2 copies, to AIME by July 1, 1953. Manuscript, Aug. 20, 1952; revised, Dec. 30, 1952. Los Angeles Meeting, February 1953.

Free Energy Diagram

In Fig. 1, the standard free energies of vaporization per mol (ΔF°_T) have been plotted for the temperature range 0° to 2000°C for the reaction

$$\text{M (solid or liquid)} = \text{M (gas)}$$

The data have been obtained from the free energy functions $-(F\text{-}H_{298})/T$ for the solid, liquid, and gaseous elements and the heats of vaporization at 298°K, i.e., ΔH_{298} (vapor). The values of these functions have been taken without change from the recent critical survey by Brewer;[5] the bulk of this data is from Kelley[7] with adjustments in the light of more recent work. Values of the function $-(F\text{-}H_{298})/T$ at T = 298°, 500°, 1000°, 1500°, and 2000°K were employed. Where possible, a further independent value was obtained by using the boiling point, i.e., where $\Delta F^\circ_T = 0$. All calculations are based on the assumption that the metal vapors are monatomic and that the boiling points refer to those temperatures at which the partial pressure of the monatomic vapor above the metal is 760 mm. For the alkali metals, e.g., sodium, potassium, and lithium, diatomic gas molecules are present as minor species; details of these are discussed by Brewer.[5]

On the basis of the above values and allowing for the change in slope at the melting point, straight line graphs were constructed and extrapolated to 2000°C. The deviation from a straight line did not exceed ±500 cal, which in all cases appears less than the probable error in the ΔF°_T values.

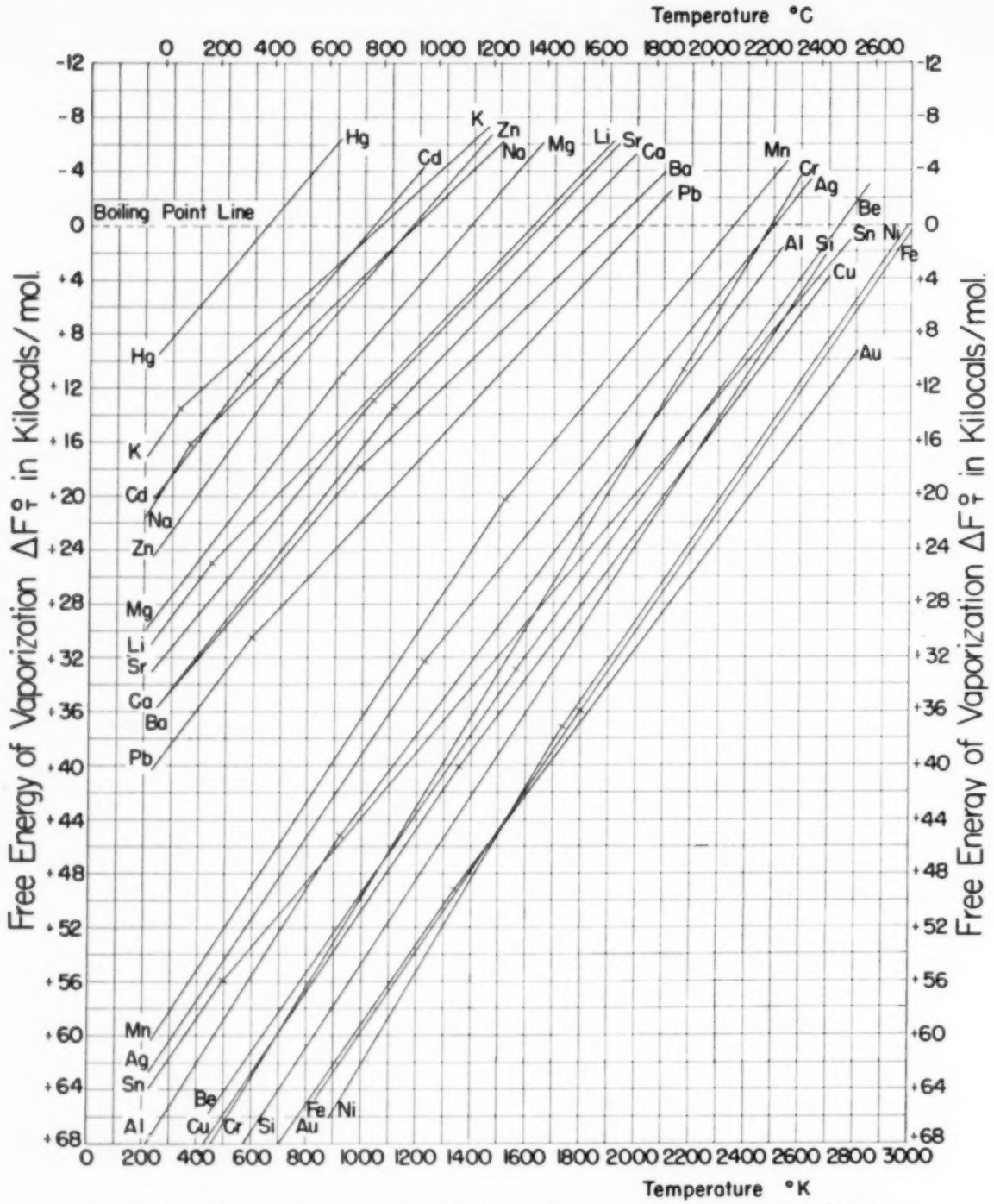

Fig. 1—Free energies of vaporization of solid and liquid elements (ΔF°_{T}). Break in line occurs at melting point.

It will be noted that the slopes of the free energy lines are approximately the same; this slope corresponds to the temperature dependent term in the expression $\Delta F = \Delta H - T\Delta S$, i.e., the entropy change which, as it involves the simple physical change from liquid or solid to monatomic gas, would be approximately the same for all metals. The relative positions of the free energy lines and, hence, of the volatilities are determined by the heats of vaporization which clearly vary considerably from one metal to another. For metals boiling below about 2000°K, the heats of vaporization are probably accurate to within one or two kilocalories per mol, but for the higher boiling metals, the heats of vaporization may be in error by ±5 kcal or more.

Use of the Diagram

To those familiar with other free energy diagrams, the application of this present diagram will be fairly clear. One example, however, will be given by con-

sidering the thermal reduction of barium oxide with aluminum at 1500°K, i.e., 1227°C, and calculating the equilibrium barium gas pressure over this reaction.

The essential reaction is

$$3BaO + 2Al(liquid) = 3Ba(gas) + Al_2O_3 \quad [1]$$

From Osborn's data[4]

(a) $3BaO = 3Ba(liquid) + 3/2O_2$ $\Delta F^\circ_{1500} = +294$ kcal

(b) $2Al(liquid) + 3/2O_2 = Al_2O_3$ $\Delta F^\circ_{1500} = -288$ kcal

From Fig. 1 (c) $3Ba(liquid) = 3Ba(gas)$ $\Delta F^\circ_{1500} = +24$ kcal

Adding reactions (a), (b), and (c) gives Eq. 1 for which

$$\Delta F^\circ_{1500} = +30{,}000 \text{ cal}$$

The equilibrium constant for Eq. 1 is given simply by $K = p^3_{Ba}$ if it is assumed that there is no interaction between barium and aluminum and that liquid aluminum, barium oxide, and alumina are at unit activities.

By definition, $\Delta F^\circ_T = -RT \ln K$

so that $\Delta F^\circ_T = -3RT \ln p_{Ba}$

By substituting $\Delta F^\circ_{1500} = +30{,}000$ cal, $T = 1500°K$, and $R = 1.98$ cal, the calculated equilibrium vapor pressure of barium over Eq. 1 is about 27 mm Hg, which is a suitable pressure for the vacuum production of the metal commercially.

Vapor Pressures

Although the main purpose of the diagram is for free energy calculations, the diagram shows at a glance the relative volatilities over the whole temperature range, those with the greatest volatility having the lowest positive free energy change. The actual vapor pressure in atmospheres is related to the free energy of vaporization by the formula

$$\Delta F^\circ_T = -RT \ln p$$

This may be conveniently written:

$$\log_{10} p \text{ (mm Hg)} = \frac{-\Delta F^\circ_T}{4.57\,T} + 2.88 \quad [2]$$

Eq. 2 gives values for the vapor pressures which agree quite well (within 10 to 20 pct) with the three or four term equations usually quoted in the literature. For the more volatile metals, however, e.g., sodium and potassium, quite small errors in the ΔF°_T values of the order of only 500 cal give rise to substantial errors in the calculated vapor pressures. Where the ΔF°_T value is less than about +5000 cal, the calculated vapor pressure as obtained from Eq. 2 should be used with caution and the use of the more accurate multiterm equation is recommended.[6]

Acknowledgment

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Technical Note

Simple Devices for Approximating Constant Stress During Tensile Creep Tests

by R. L. Fullman, R. P. Carreker, Jr., and J. C. Fisher

CREEP tests are usually constant load tests. As deformation occurs, the cross section of the specimen changes so as to maintain approximately constant volume. Until necking starts,

$$\frac{A}{A_o} \cong \frac{l_o}{l}$$

where A is the cross-sectional area, l is the specimen length, and the subscript o indicates initial values. Thus, during a tensile creep test under constant load, the stress increases in proportion to the elongation.

$$S = \frac{F}{A} \cong \frac{F}{A_o}\frac{l}{l_o}$$

where S is the stress and F the load applied to the specimen.

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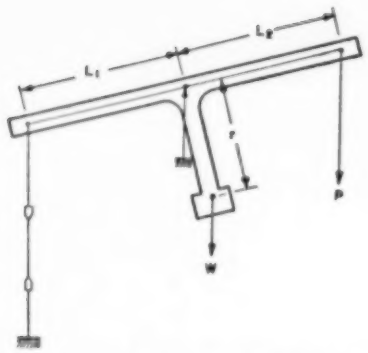

Fig. 1—Schematic diagram of loading beam with center of gravity below fulcrum.

It has been repeatedly emphasized in the literature that, while constant load tests are convenient and may approximate some types of service conditions, constant stress tests are needed to gain an understanding of the nature of the deformation process. Constant load creep behavior can be calculated from constant stress data, but the reverse is much more difficult.

Andrade has described three methods of maintaining constant stress during creep tests,[1-3] none of which is as simple, from an equipment standpoint,

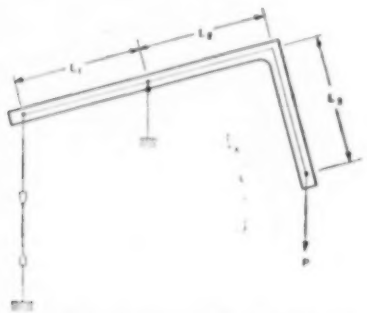

Fig. 2—Schematic diagram of loading beam with knife edges not collinear.

as the usual constant load test. Fisher and Carreker have described a method which maintains an approximately constant stress with relatively simple equipment.[4] The method described below maintains a more nearly constant stress than that of Fisher and Carreker, while retaining the simplicity of the constant load test. The method requires only slight modification of most constant-load creep testing machines employing a beam loading arrangement. The mechanical advantage of the beam loading device is retained, permitting large stresses to be applied with relatively small weights.

Within the authors' knowledge, the method to be described originated in their joint discussion.[5] It has been used successfully in their laboratory for several years and is now being used in other laboratories. Its simplicity and utility seem to warrant description.

If the fulcrum of a beam does not lie on a straight line containing the other knife-edges and the beam's center of gravity, the force transmitted by the beam is a function of the angle between the beam and the horizontal. When approximately horizontal, a simple beam having its fulcrum in a straight line containing the other knife-edges will maintain nearly constant stress in a tensile specimen if the center of gravity is located a distance r directly below the fulcrum, such that

$$r = \frac{P L_1 L_2}{W l_1} \qquad [1]$$

where P is the applied load, L_1 and L_2 are the beam dimensions as shown in Fig. 1, W is the weight of the beam, and l_1 is the specimen length when the beam is horizontal. For optimum results, the initial angle of the beam should be such that it reaches a horizontal position when the specimen has attained half the maximum elongation of the test. Constant

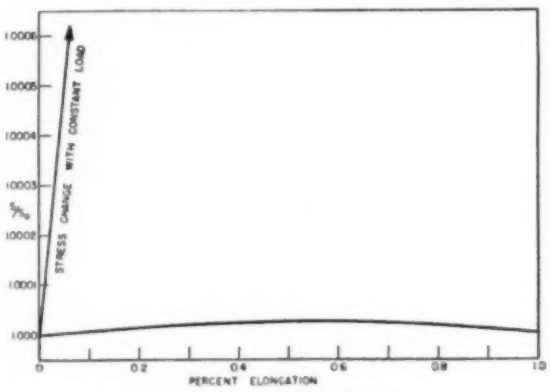

Fig. 3—Stress change in tensile creep test.

stress is not maintained after the specimen starts to neck.

Existing creep machines employing a loading beam may be modified to make use of this method by a change in position of the beam center of gravity. With some equipment this change may be accomplished by the adjustment of existing counterweights. However, the method is somewhat inconvenient in that the position of the center of gravity or the beam weight must be changed for every change in load.

A more convenient application of the same principle may be obtained with a slight modification of the beam. In this case the center of gravity of the beam is placed at the fulcrum, but the fulcrum is not located on a straight line containing the other knife-

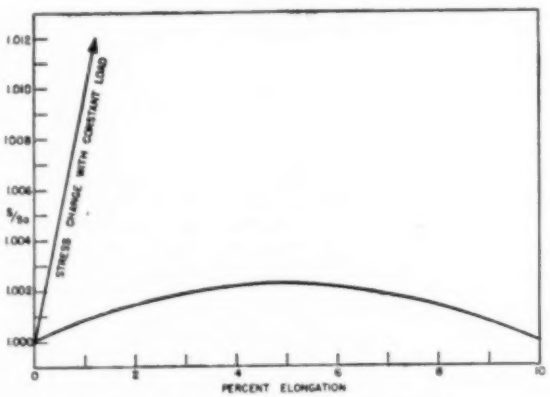

Fig. 4—Stress change in tensile creep test.

edges. Regardless of load, when the beam is approximately horizontal, a tensile specimen is deformed under nearly constant stress if

$$L_s = \frac{L_1 L_2}{l_1} \qquad [2]$$

where l_1 is the specimen length when the beam is horizontal, and L_1, L_2, and L_s are the beam dimensions, as shown in Fig. 2. For optimum results, the initial angle of the beam should again be such that it reaches a horizontal position when the specimen is at half the maximum elongation of the test.

Figs. 3 and 4 show the variation of stress as a function of strain for maximum elongations of 1 and

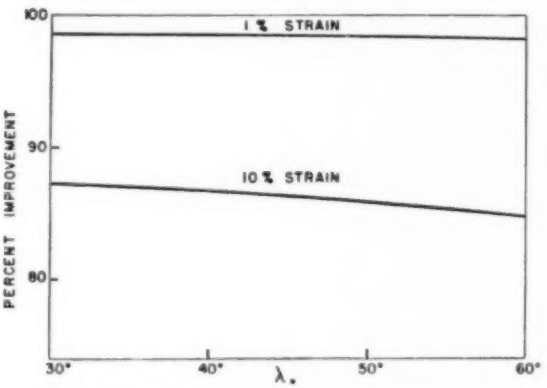

Fig. 5—Influence of crystal orientation on constancy of resolved shear stress.

10 pct, if $L_1 = L_2 = L_s = l_1$. The same result is obtained using a beam with a low center of gravity, if $W = P$ and $r = L_1 = L_2 = l_1$. The efficiency of the method can be expressed in terms of "percentage improvement" in constancy of applied stress, defined as

$$\left(\frac{\Delta S_1 - \Delta S_2}{\Delta S_1}\right) \times 100 \text{ pct}$$

Here ΔS_2 is the maximum stress change using the type of beam described and ΔS_1 is the maximum stress change with a constant load. For the case considered in Figs. 3 and 4, the percentages of improvement are: for 1 pct total elongation, 99.75 pct; and for 10 pct total elongation, 97.76 pct. A different choice of dimensions causes only a slight change in the efficiency of the method, provided Eq. 1 or Eq. 2 is satisfied. For any choice of dimensions which satisfies one of these equations,

$$S = \frac{Pl}{A_o l_o}\left[\frac{L_2}{L_1} + \frac{L_2 (l_1 - l)}{l_1 \sqrt{L_1^2 - (l_1 - l)^2}}\right] \qquad [3]$$

where l_o and l are initial and other values of specimen length, and A_o is the initial cross-sectional area of the specimen.

In tensile creep testing of single crystal specimens, it is desirable to maintain at a constant value the shear stress component on the active slip plane and in the active slip direction.* By suitable choice of the beam dimensions, the loading beam just described can be used to maintain approximately constant resolved shear stress during creep of single crystals.

* Hereafter this stress component will be called simply the "resolved shear stress."

The resolved shear stress, RSS, is given by the relationship[6]

$$\text{RSS} = \frac{F}{A} \sin \chi \cos \lambda$$

where χ is the angle between the tensile axis and the active slip plane and λ is the angle between the tensile axis and the active slip direction. During deformation by single slip, the crystal orientation and shape change in such a manner that

$$\frac{l}{l_o} = \frac{A_o}{A} = \frac{\sin \chi_o}{\sin \chi} = \frac{\sin \lambda_o}{\sin \lambda} \qquad [4]$$

Combining this relationship with the geometry of the loading beam shown in Fig. 2, it is found that the resolved shear stress is maintained at approximately constant value if

$$L_s = \frac{L_1 L_2}{l_1} \tan^2 \lambda_1 \qquad [5]$$

where λ_1 is the value of λ at half the maximum strain of the test (i.e., the value of λ at $l = l_1$, as computed from Eq. 4). The resolved shear stress will then change during deformation in accordance with

$$\text{RSS} = \frac{P \sin \chi_o}{A_o L_1}\left(L_2 - \frac{L_s (l - l_1)}{\sqrt{L_1^2 - (l - l_1)^2}}\right)\left(\frac{\sqrt{l^2 - l_o^2 \sin^2 \lambda_o}}{l}\right) \qquad [6]$$

Fig. 5 shows the influence of initial orientation of the crystal upon "percentage improvement" in constancy of resolved shear stress, defined by

$$\left(\frac{\Delta S_1 - \Delta S_2}{\Delta S_1}\right) \times 100 \text{ pct}$$

Here ΔS_1 is the maximum change in resolved shear stress during a constant load test and ΔS_2 is the maximum change in resolved shear stress during a test using a loading beam with dimensions that satisfy Eq. 5. The curves of Fig. 5 were computed for strains of 1 and 10 pct, and with $L_1 = L_2 = l_1$. Other dimensions would lead to similar results.

Conclusions

Proper choice of dimensions of a simple loading beam permit maintenance of very nearly constant stress during creep testing of polycrystalline materials or of nearly constant resolved shear stress during creep testing of single crystals.

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Electron Microscope Study of the Effect of Cold Work On the Subgrain Structure of Copper

by Laurence Delisle

This work represents the first step of an attempt to test the applicability of the electron microscope to the study of subgrain structures in copper. Observations on annealed and deformed single crystals and polycrystalline samples of copper are described.

IN the course of study of the structure of fine tungsten wires and tungsten rods with the electron microscope, well defined subgrain structures were observed. The size, size distribution, and orientation uniformity of the etch figures varied widely in different samples. Figs. 1 and 2, electron micrographs of a tungsten wire and of a tungsten rod, respectively, are illustrations of the difference in size and size distribution of the etch figures in different samples of the same metal. The observed differences, as pointed out in a previous paper,[1] appeared to be related to the heat and mechanical treatments of the samples. They were also consistent with the results reported in the literature on the mosaic structure of metals.[2] For that reason a program of research was initiated in an effort to obtain more systematic evidence of the possible relation of heat and mechanical treatments to the subgrain structure of metals as observed in the electron microscope. The purpose of this paper is to present observations made on the effect of cold work on the subgrain structure of copper.

Procedure

Starting Materials: Copper was the metal studied because it can be obtained in a high degree of purity, much information is available in the literature on its properties and its response to cold work and heat treatment, it shows no allotropic change, and it is sufficiently hard to be handled without great difficulty. Two groups of specimens were used: 1—single crystals cast from spectroscopically pure copper and 2—polycrystalline samples of oxygen-free high conductivity copper. Single crystals were studied because it was hoped that the elimination of a number of variables, such as grain boundaries, orientation differences, degree of purity, would simplify the problem and perhaps permit a better understanding of the phenomena that would be observed. The polycrystalline samples were designed to give a general picture of the changes considered.

The single crystals were made of copper which analyzed spectroscopically to better than 99.999 pct Cu. They were cast in vacuum, by the Bridgman method, in crucibles made of graphite with a maximum ash content of 0.06 pct. The mold design is shown in Fig. 3. It permitted casting crystals of the size and shape required for the experiments, so that the danger of introducing cold work in the original samples by cutting or other machining would be eliminated.

The polycrystalline samples were pieces, ¾ in. long, cut from a rod of oxygen-free high conductivity copper, ⅜ in. in diameter. A flat surface, ¼ in. wide, was milled along the rods, polished, and etched. The samples were then annealed in vacuum at 850°C for 1 hr.

Polishing and Etching: Work previously done on tungsten,[1] polished mechanically and etched chemically,* had shown that: 1—the general appearance of the etch figures of a given sample was not altered by repeated polishings and etchings under similar conditions; 2—variations in the time of etching and the concentration of the etchant changed the definition of the etch figures, but did not alter their general size nor orientation distribution within the limits of observation. Further work confirmed the reproducibility of the subgrain structures observed in, 1—single crystals and polycrystalline samples of copper when polishing and etching were repeated under similar conditions, and 2—specimens of tungsten and polycrystalline copper when electrolytic polishing and etching were substituted for mechanical polishing and chemical etching, respectively. On the strength of these observations, it was felt that, if conditions of polishing and etching were kept constant, changes observed in the subgrain structure of a sample upon deformation and annealing would be attributable to such treatments. For that reason the conditions of polishing and etching were kept as constant as possible.

The single crystals were polished electrolytically in a bath of orthophosphoric acid in water, in the ratio of 1000 g of acid of density 1.75 g per cc to 1000 cc of solution, under a potential drop of 1.6 to 1.8 v. Electrolytic polishing was selected to prevent the formation of distorted metal in polishing. The same samples were etched by immersion in a 10 pct aque-

* By "chemical etch" is meant attack of the surface of the specimen simply by flooding with or immersion into the etchant..

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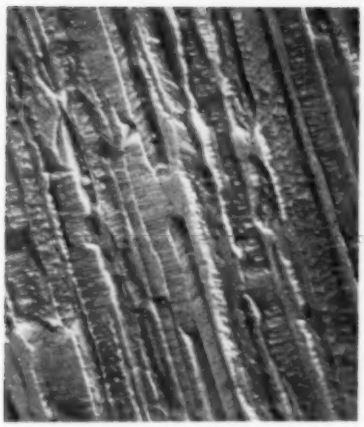

Fig. 1 (left)—Longitudinal section from tungsten wire 0.003 in. in diam. X10,000. Area reduced approximately 50 pct for reproduction.

Fig. 2 (right)—Tungsten rod. X10,000. Area reduced approximately 50 pct for reproduction.

ous solution of $(NH_4)_2S_2O_8$. They required a long etching time, of the order of 1 hr. As was already observed in the case of tungsten, shorter etching time brought out poorly defined structures. As the etching time was increased, the structure gradually became clear. At such a point a steady state of solution was approached since no changes could be observed upon prolonging the etching time by as long as 1 hr. For that reason the sharpness of the structures, rather than the etching time, was used as a guide in etching the single crystals.

Table I. Amount of Compression Applied to Specimens

Specimen Designation	Degree of Compression, Pct
Single Crystals	
S0	17.0
S1	20.0
Polycrystalline Samples	
PO1	15.0
PO2	23.0
P1	12.4
P2	22.0

The polycrystalline samples were polished mechanically and etched for 5 min by immersion in a 10 pct aqueous solution of $(NH_4)_2S_2O_8$. Mechanical polishing was chosen because it permits a better control of the flatness of the surface. The polishing of the original machined samples was carried out on emery paper, through No. 40, and on silk and velvet wheels. The specimens were repolished after the 850°C anneal prior to deformation. Polishing in this case, as well as polishing subsequent to deformation, was done only on silk and velvet. In both cases, alternate final polishing on velvet and etching were repeated three times.

Deformation: The specimens were deformed in compression. The strains produced are given in Table I.

Examination with the Electron Microscope: Parlodion replicas, stripped dry from the surface of the specimens and shadow cast with manganese were the specimens observed in the electron microscope.

Prior to examination with the electron microscope, polishing and etching were controlled as far as possible with the light microscope. Within the limit of resolution of the light microscope, there was always agreement between the structures observed in the light and in the electron microscopes.

Observations on Single Crystals

Fig. 4 is representative of the structure of one of the single crystals as cast and etched. The structure consists of blocks with edges that are practically straight and faces that appear smooth. Fig. 5, a stereographic projection of the crystal, obtained from an X-ray back-reflection pattern by the Greninger method, shows that a {111} plane is nearly parallel to the plane of Fig. 4, an orientation consistent with the three-fold symmetry exhibited by triangular areas between the blocks shown in the micrograph. Goniometric measurements indicated that etching took place on {110} and {111} faces.

Slip lines appear on the surface of the single crystals after deformation (Figs. 6, 8, and 9). The spacing and intensity of the slip lines that could be observed are not uniform. The closest visible spacing of the flow lines is of the order of 0.1 micron in the plane of the figure. A fine structure is also visible on the surface of some of the blocks. It shows a parallel arrangement along lines closer than 0.1 micron, suggestive of slip lines. Upon repolishing and etching the deformed single crystals, a block structure appears as in the cast samples. Within these larger blocks, however, the fine structure which was faintly visible before repolishing and etching is clearly brought out, Fig. 7. After deformation, the edges of the larger blocks are neither as straight nor as smooth as they appeared in the cast single crystals.

Observations on Polycrystalline Samples

Figs. 10 and 11 illustrate the structure after repolishing and etching, of the polycrystalline specimens annealed for 1 hr at 850°C. Within the grains a block structure appears, but the blocks are smaller than those observed in the single crystals. As will be shown further, by comparison with the subgrain structure in deformed samples, the subgrain blocks of the fully annealed specimens are uniform in size and orientation, and that uniformity extends from

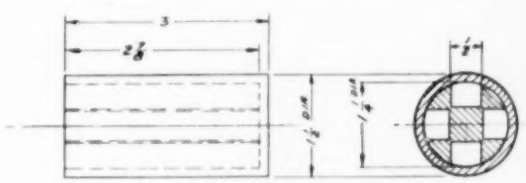

Fig. 3—Graphite mold used to cast single crystals.

Fig. 4—Copper single crystal S1, as cast and etched. X4000. Area reduced approximately 50 pct for reproduction.

the center to the boundaries of the grains. In Fig. 12 the blocks within a grain are shown. As in the case of the cast single crystals, the blocks have edges that are nearly straight lines and the faces of most of them show no structural details.

As a result of deformation, slip lines appeared on the surface of the specimens. As is to be expected, the direction of the slip lines changes with the orientation of the grains, Figs. 13 and 14. In some grains, a multiplicity of slip lines indicates that several slip systems were operative, Fig. 15. The slip lines that can be seen within a grain are not evenly spaced, and are often crowded at grain boundaries, Fig. 16. The slip lines within a grain stop at the grain boundaries, Figs. 16 to 18, and slip lines in a different direction are sometimes visible in adjacent grains. In Figs. 16 and 17, one set of slip lines appear clearly in one of the grains and a second set, marked F, is faintly visible in the adjacent grain.

Upon repolishing and etching the polycrystalline samples, the slip lines disappear, but other evidences of deformation remain or become visible:

1—Some of the subgrain blocks are severely deformed, especially near grain boundaries, Figs. 19 and 20.

2—The uniformity of orientation of the subgrain blocks that was observed within most grains of the annealed material is destroyed by cold work, Figs. 21 to 23. Within grains, cold work seems to produce shifts in orientation. Such shifts manifest themselves by a change in the appearance of the subgrain blocks that look as if they were seen from different angles in different parts of the grains. The orientation shift is visible across Fig. 21 from the bottom left-hand corner to the top right-hand corner. A comparison of Figs. 10 and 21 brings out the difference in orientation uniformity before and after deformation, respectively. The change in orientation produced by cold work within individual grains is even more pronounced at grain boundaries, Fig. 22. A comparison of Figs. 22 and 18 suggests that the regions of a deformed grain that appear differently oriented after repolishing and etching may be bounded by glide planes.

3—A structure, much finer than the original blocks, of the order of 1000A, appears, after cold working, at grain boundaries, on the surface of the blocks that had a smooth surface after annealing, and often in the angles formed by meeting subgrain blocks, Figs. 24 and 25. This fine structure is of the same order of size as the fine structure observed in deformed single crystals after electrolytic repolishing and etching, Fig. 7.

4—Some subgrain blocks broke completely into lamellae, Fig. 26. The subgrain blocks shown in the upper and in the lower parts of Fig. 26 are differently oriented within the same grain. They are separated by a region of blocks consisting of lamellae. The arrangement suggests that rotation of the lamellae of the intermediate blocks permits a fit between the two regions of differently oriented blocks of the same grain.

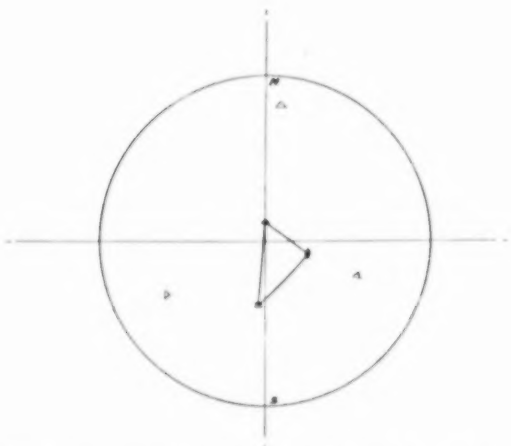

Fig. 5—Stereographic projection of single crystal S1, as cast.

Fig. 6 (left)—Copper single crystal S1, deformed 20 pct. No repolishing nor etching after deformation. X4000. Area reduced approximately 50 pct for reproduction.

Fig. 7 (right)—Copper single crystal S1, deformed 20 pct. Repolished and etched after deformation. X4000. Area reduced approximately 50 pct for reproduction.

Fig. 8 (left)—Copper single crystal S1, deformed 20 pct. Face other than that shown in Fig. 6. No repolishing nor etching after deformation. X10,000. Area reduced approximately 50 pct for reproduction.

Fig. 9 (right) — Copper single crystal SO, deformed 17 pct. No repolishing nor etching after deformation. X10,000. Area reduced approximately 50 pct for reproduction.

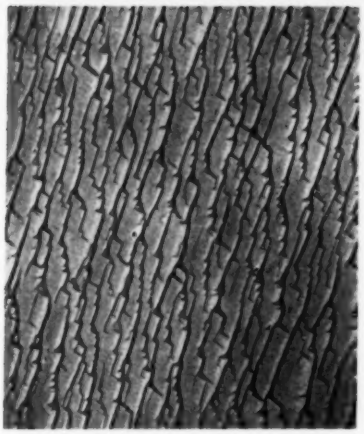

Fig. 10 (left)—Polycrystalline copper, annealed, polished, and etched. Blocks within a grain. X5000. Area reduced approximately 50 pct for reproduction.

Fig. 11 (right)—Polycrystalline copper, annealed, polished, and etched. Blocks in adjacent grains. X5000. Area reduced approximately 50 pct for reproduction.

Fig. 12 (left)—Polycrystalline copper, annealed, polished, and etched. Blocks within a grain. X10,000. Area reduced approximately 50 pct for reproduction.

Fig. 13 (right)—Polycrystalline copper, deformed. No repolishing nor etching after deformation. X10,000. Area reduced approximately 50 pct for reproduction.

Fig. 14 (left)—Polycrystalline copper, deformed. No repolishing nor etching after deformation. X10,000. Area reduced approximately 50 pct for reproduction.

Fig. 15 (right)—Polycrystalline copper, deformed. No repolishing nor etching after deformation. X10,000. Area reduced approximately 50 pct for reproduction.

Fig. 16—Polycrystalline copper, deformed. No repolishing nor etching after deformation. X10,000. Area reduced approximately 50 pct for reproduction.

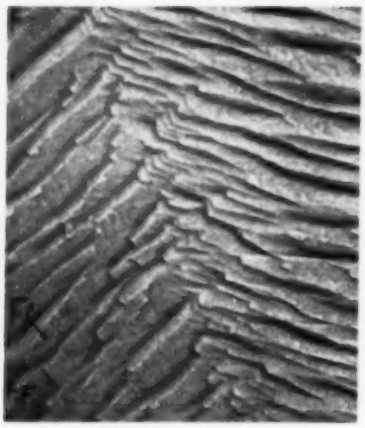

Fig. 17—Polycrystalline copper, deformed. No repolishing nor etching after deformation. X10,000. Area reduced approximately 50 pct for reproduction.

Fig. 18—Polycrystalline copper, deformed. No repolishing nor etching after deformation. X10,000. Area reduced approximately 50 pct for reproduction.

Fig. 19—Polycrystalline copper, deformed 22 pct. Polished and etched after deformation. X10,000. Area reduced approximately 50 pct for reproduction.

Fig. 20—Polycrystalline copper, deformed 22 pct. Polished and etched after deformation. X10,000. Area reduced approximately 50 pct for reproduction.

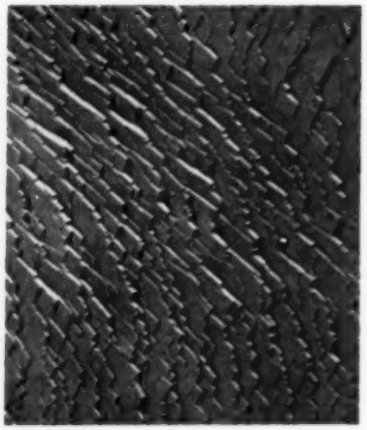

Fig. 21—Polycrystalline copper, deformed. Polished and etched after deformation. X5000. Area reduced approximately 50 pct for reproduction.

Fig. 22—Polycrystalline copper, deformed. Polished and etched after deformation. X5000. Area reduced approximately 50 pct for reproduction.

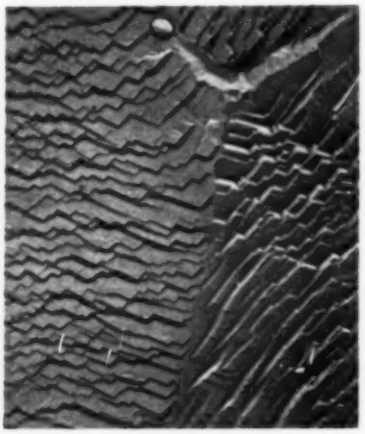

Fig. 23—Polycrystalline copper, deformed. Polished and etched after deformation. X5000. Area reduced approximately 50 pct for reproduction.

Fig. 24—Polycrystalline copper, deformed 22 pct. Repolished and etched after deformation. X10,000. Area reduced approximately 50 pct for reproduction.

Discussion of Results

The electron micrographs presented in this paper add support to the existence of a complex subgrain structure in metals. The size of the smallest subgrain units observed in copper, like those previously observed in tungsten,[1] of the order of 1000A, corresponds to that calculated by Bragg, Darwin, and James[3] for the units of a mosaic structure from measurements of intensity of reflection of X-rays by crystals, by Wood[4] for the limiting size of crystallites from line broadening, and by Born[5] for a perfect lattice from quantum mechanics considerations. Such a dimension was used by Bragg[6] to compute a satisfactory value for the elastic limit of strain of many metals. The size of the larger units of the subgrain structures observed in copper in this paper, and in tungsten previously,[1] is of the order of magnitude calculated by Taylor[2] for the length through which a dislocation travels before being stopped by a surface of misfit. There seems to be therefore a size relationship between the subgrain structures observed with the electron microscope and the mosaic structures proposed by a number of authors.

On the other hand, there appears to be a relationship between mosaic structures and dislocations. According to Cottrell,[9] it is "reasonable to suppose that the more macroscopic irregularities in crystals, e.g. mosaic structure and crystal fragmentation in the cold work state, are built up from assemblies of dislocations." If dislocations exist, the metal in the vicinity of a dislocation is stressed and for that reason its rate of etching should be faster than that of the stress-free material. If the blocks of the subgrain structure are considered as separated by arrays of dislocations, they should be brought out by etching. Read and Shockley[10] explain that veining obtained by Lacombe[11] is the result of etching that starts at dislocations. Once started, the etching proceeds on the planes that dissolve fastest. The same planes have slightly different orientations in different blocks and that difference accentuates the effect of etching. On a different scale of magnitude and disorientation, the phenomenon is similar to the etching of grains as ordinarily applied in metallography. The sizes of the subgrain blocks in the cast single crystals and in the annealed polycrystalline samples of copper studied in this paper agree with the calculations of Read and Shockley[10] for small angles of misfit.

The number of dislocations should be a function of internal stresses and degree of purity.[7, 8, 33-35] The single crystals used in these experiments were made of spectroscopically pure copper, they were cast in graphite of high purity, in vacuum, and they were cooled very slowly. They were therefore freer from impurities and internal stresses than the polycrystalline samples, made of commercial copper, and they should have a lower density of dislocations. The subgrain blocks of the cast single crystals are indeed much larger and therefore seem to represent a higher degree of perfection than those of the polycrystalline samples, Figs. 4 and 11.

The number of dislocations should be increased by cold work. The appearance of a fine structure within the blocks of the subgrain structure of the single crystals and the polycrystalline samples upon deformation, Figs. 7, 25, 26, could be explained by an increase in the number of dislocations. In Fig. 7, the alignment of the fine structure within the larger blocks suggests traces of slip planes. Such a structure was not visible in the cast crystals. It seems therefore that it was brought out by cold work. It

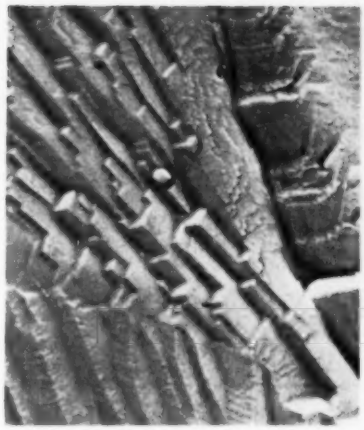

Fig. 25—Polycrystalline copper, deformed 22 pct. Repolished and etched after deformation. X10,000. Area reduced approximately 50 pct for reproduction.

is visible after etching, so that it is not a form of slip lines. It could be a structure that existed in the cast crystals,[5, 6] but could not be resolved because the degree of disorientation of its units was too small. Deformation increased the degree of disorientation and permitted differential etching. It could also be recrystallized crystallites formed, according to Wood,[4] when cold work on a slip system reaches a certain value. This fine structure has the characteristics of the strain markings reported by Burke and Barrett.[35] If the evidence produced by Burke and Barrett that the strain markings are traces of slip planes rather than mechanical twins is accepted, the etching of such markings could be the result of the presence of the recrystallized crystallites proposed by Wood.

The individual subgrain blocks of the cold-worked samples, after repolishing and etching, often appear bent by the deformation, Figs. 19 and 20. Their size is comparable to the size they had before deformation, Figs. 12 and 20, and they can easily be visualized as the original blocks strained individually by different amounts in different regions of the grains. Such a mechanism seems possible only if the subgrain blocks revealed by etching are a real structure like the grains themselves.

It has been proposed by Burgers,[36] Kochendorfer,[37] and Laurent[38] that dislocations during cold work are created at crystallite boundaries which produce notch effects. The stress field of the new dislocations counteracts the notch effect so that the production of new dislocations becomes more difficult and strain hardening occurs. In the single crystals, the more clearly visible flow lines often pass through the angles formed

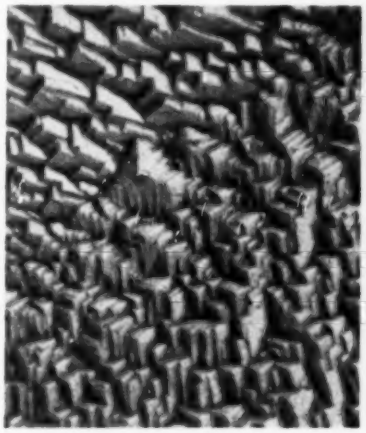

Fig. 26—Polycrystalline copper, deformed 15 pct. Repolished and etched after deformation. X10,000. Area reduced approximately 50 pct for reproduction.

by meeting subgrain boundaries that could act as notches, Fig. 9. When several slip systems are operative, the dislocations connected with the first slip system should be an obstacle to the motion of the dislocations on a second slip system. In Fig. 15, the prominent, more widely spaced slip lines appear to have been stopped at places by a finer structure. In the single crystals, no grain and fewer subgrain boundaries hindered slip than in the polycrystalline samples. For that reason perhaps deformation proceeded in one slip system more extensively and slip lines in single crystals are often found to be closely spaced, Figs. 6 and 8. The latter figures are reminiscent, although on a different magnification scale, of electron micrographs of slip lines in aluminum published by Heidenreich and Shockley.[19]

On account of the glide in different directions, a slip plane becomes a grid of dislocations which, according to Cottrell,[7] "constitutes a transition surface between two crystals rotated with respect to each other about an axis normal to the transition surface." The formation of such transition surfaces would explain the presence, upon deformation, of regions differently oriented within individual grains of the polycrystalline samples, Figs 21 to 23 and 26.

This paper described part of the results of an attempt to apply the electron microscope to the study of the changes in the subgrain structure of metals produced by cold work and recovery. Because its scope was wide and it represented a relatively new method of approach, the work was intended to be in the nature of a survey. For that reason no claim was made at conclusiveness and only possible relationships between accredited theories and the structures observed were indicated. It is hoped, however, that the possibility of the relationships pointed out will be considered sufficient to arouse interest in the potentialities of the electron microscope for the study of subgrain structures in metals.

Conclusions

1—A subgrain structure was observed, by means of the electron microscope, in single crystals of spectroscopically pure copper and in polycrystalline samples of oxygen-free high conductivity copper.

2—A study was undertaken in an effort to find out whether consistent changes would be produced by cold work and recovery in the subgrain structures observed in copper. Observations on the changes produced by cold work are reported.

3—The subgrain blocks observed in the cast and slowly cooled single crystals, as well as in the annealed polycrystalline samples, appear to have relatively straight edges and smooth surfaces.

4—The subgrain blocks observed in the cast single crystals are larger than those observed in the annealed polycrystalline samples.

5—Repolishing and etching of the deformed single crystals and polycrystalline samples brought out, within blocks that appear to correspond to the subgrain blocks observed prior to deformation, a finer structure, the size of which is of the order of 1000Å.

6—The degree of disorientation of the larger subgrain blocks within a single crystal or individual grains of the polycrystalline samples is increased by cold work. The larger subgrain blocks also appear individually deformed by cold work.

7—The larger and finer subgrain structures observed differ in size by an order of magnitude. They both correspond to crystallite dimensions reported in the literature on the basis of entirely different observations.

8—Comparable structures were previously observed in tungsten.

Acknowledgments

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Martensite Habit Plane in Quenched Ti-Mn Alloys

by You Chao Liu and Harold Margolin

Investigation of martensite habit plane in water-quenched Ti-Mn alloys was carried out in the range of manganese contents between 4.35 and 5.25 pct. On the basis of 22 measurements, the poles were observed to fall into two groups, indicating the existence of two habit planes. The Miller indices of the poles close to these two groups were found to be $<334>_\beta$ and $<344>_\beta$.

TITANIUM has an allotropic transformation at 882.1° ± 0.5°C[1] by which the high temperature β (body-centered cubic) phase changes to α (hexagonal close-packed) phase. It is well known, as in the case of low carbon steel, that the unalloyed high temperature β phase cannot be retained no matter how rapid the cooling rate. However, the addition of alloying elements usually lowers the transformation temperature continuously and stabilizes the β phase to a temperature much lower than the allotropic transformation temperature in pure titanium. A typical example of this behavior is found with manganese-alloying of titanium.

Fig. 1 shows the titanium-rich Ti-Mn equilibrium diagram as established by Battelle Memorial Institute.[2] The addition of manganese lowers the β transus line and the eutectoid reaction takes place at 550°C, the eutectoid composition being 20 wt pct Mn. However, upon quenching to room temperature from the β field, the β phase can be completely retained with much less manganese, see Fig. 2. If the composition is less than that required to stabilize the β phase completely, a plate-like structure, similar to that of martensite in other alloy systems, appears as shown in Fig. 3. Much lower alloying content merely intensifies the amounts of these transformed structures, as shown in Fig. 4.

This paper presents the results of an investigation of the habit plane of these martensitic structures in Ti-Mn alloys.

Experimental Procedure

Investigation of the pole of a given platelet was made using the two-surface method of analysis.[3] In order that a given martensite trace on two surfaces be readily recognized both at low and high magnifications, it is desirable that only a small amount of martensite be present. From a review of the data obtained at Battelle Memorial Institute[2] on quench hardening in Ti-Mn alloys and the results of Duwez[4] on transformation in Ti-Mo alloys, it was decided that the most appropriate Ti-Mn composition for this study would be about 6 pct Mn. A group of alloys containing from 3 to 10 pct Mn was made to bracket this composition. The choice of the 6 pct alloy was apparently proper, since it was later learned that approximately 5.5 to 6 pct Mn is required to stabilize the β phase on water quenching.

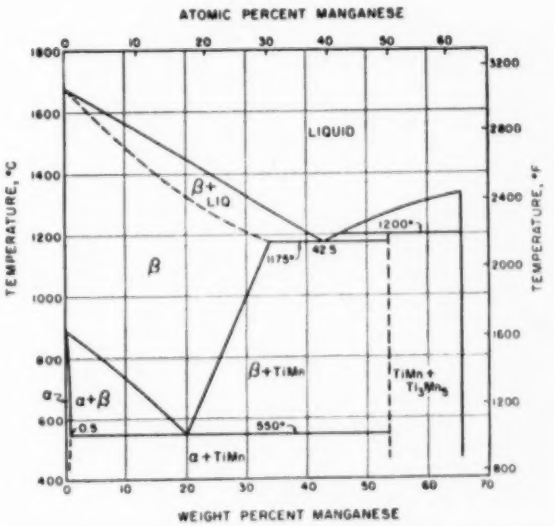

Fig. 1—Titanium-rich Ti-Mn equilibrium diagram. After Battelle Memorial Institute.

Y. C. LIU, Student Associate AIME, and H. MARGOLIN, Junior Member AIME, are Research Associates, Engineering Research Div., New York University, New York.

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Fig. 2—Ti-Mn alloy (8 pct Mn), water-quenched from 1200°C. Retained β. X75. Area reduced approximately 60 pct for reproduction.

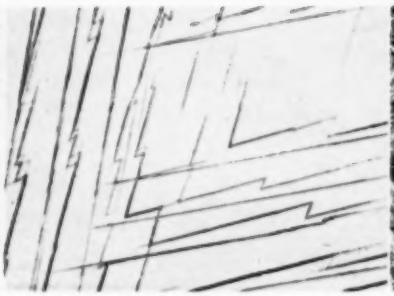

Fig. 3—Typical martensitic structure in Ti-Mn alloys in the range of compositions investigated. X150. Area reduced approximately 60 pct for reproduction.

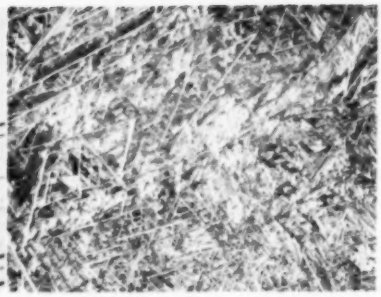

Fig. 4—Ti-Mn alloy (1 pct Mn) water-quenched from 1200°C. Transformed β. X100. Area reduced approximately 60 pct for reproduction.

Five different specimens with a nominal manganese content of 6 pct were finally used in this investigation. The actual compositions of these specimens varied from 4.35 to 5.25 pct Mn. In the case of the 4.35 pct specimen, transformation was so extensive that angular relations of selected platelets could be measured only by first taking photographs of two surfaces and then matching them. A specimen with a nominal manganese content of 3 pct produced Laue spots so broad as to make orientation determination impossible.

All specimens, $\frac{3}{8}$x$\frac{3}{8}$x$\frac{3}{4}$ in. in size, were heat-treated for 60 to 70 hr at 1200°C, in argon atmospheres to obtain grains of 0.8 to 1 cm in diameter, which proved to be adequate for this investigation.

Three polished surfaces* around the specimen corner were prepared in each specimen. This arrangement had the dual advantage that many traces across the edges were available and that the angles between polished surfaces could be measured by manipulation of the stereographic projection.

* The standard procedures for polishing and etching titanium and its alloys as employed at the New York University Metallurgical Laboratory[6] have been followed.

The stereographic method used in the measurement of the angles between surfaces is described as follows: Fig. 5 shows three polished surfaces at corner O. Angles α, β, and γ were measured with the graduated circle on a metallograph. With surface A as the plane of the stereographic projection and corner O at the center, azimuthal angle α is located as shown in Fig. 6. The intersection of planes B and C, MN, is then represented in Fig. 6 as point M in the projection. The pole of surface C (which is defined by the edges OP and MN) is designated C and is located 90° from the great circle PMP'. Pole B is similarly located and represented. The three polished surfaces, A, B, and C in Fig. 5 are thus represented in Fig. 6 by their respective poles O, B, and C. The angles between the planes are equal to 180° minus the angle between the corresponding

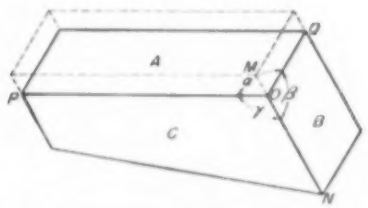

Fig. 5—Schematic diagram of intersection of surface planes A, B, and C at specimen corner.

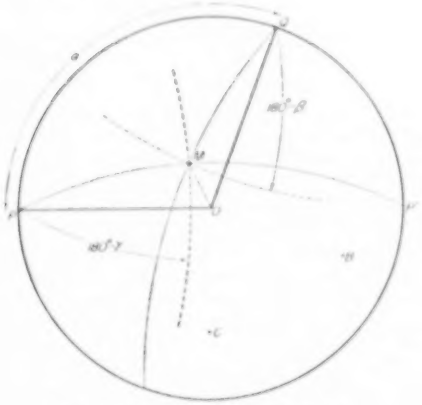

Fig. 6—Stereographic plot of Fig. 5.

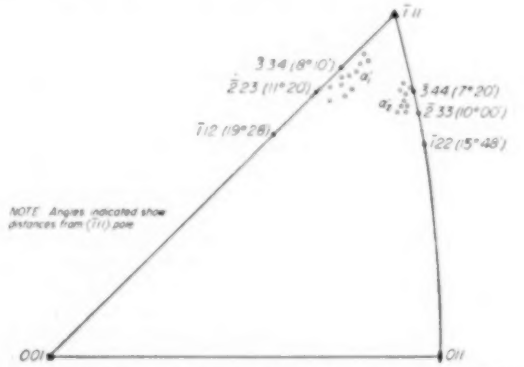

Fig. 7—Location of poles of habit planes in a standard stereographic triangle.

poles; the latter angle can be measured along great circles in the projection.

The grain orientations were determined from Laue patterns by the back-reflection X-ray method.[7] In the early stage of this work, difficulties were encountered due to the peculiar nature of the back-reflection X-ray photograms obtained. These showed high intensity spots, not related to any zones, and a scarcity of zones. The number of Laue spots was limited due to the fact that titanium has a very high absorption coefficient with respect to copper radiation. By using an exposure time of 18 hr with a current of 20 ma at 35 kv, sufficient spots for orientation determination were produced. The high in-

tensity spots could not be accounted for and were not considered. The orientations of two polished surfaces of a grain were checked against the value for the angle between these two planes as determined by the stereographic method described earlier. The difference between these two orientations, when brought into coincidence, was about one degree.

After it was learned that two habit planes occur in the present alloy system, it seemed appropriate to attempt to correlate the microstructure with habit plane. The fineness of the transformed structure in most cases ruled out the possibility of using the two-surface method; consequently a single-surface method was employed. The validity of the latter analysis then depended on the consistency of a large number of measurements.

In the single-surface analysis the orientation of the grain where the microstructure is to be studied, as shown in Fig. 8, is plotted on a stereographic projection. All permutations of the two habit planes, a total of 12 each, are therefore plotted as the area corresponding to the original data obtained, Fig. 7. The trace normal of the investigated traces is then superimposed on the projection. If the habit plane of these traces corresponds to either of the two habit planes found, the trace normals must pass through one of these 24 areas.

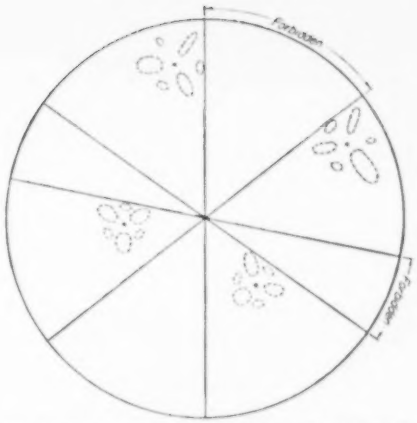

Fig. 8—Stereographic plot of single-surface analysis of traces of fine transformed needles.

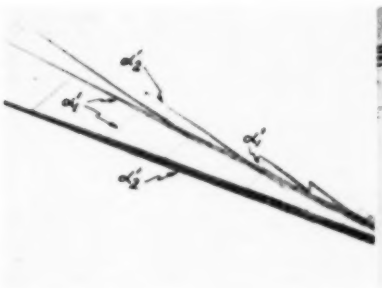

Fig. 9—Intimate association of α'_1 and α'_2 platelets. X400. Area reduced approximately 60 pct for reproduction.

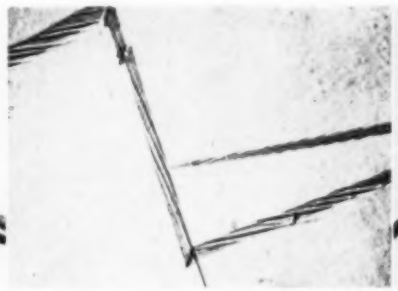

Fig. 10—"Twins" within martensite. X650. Area reduced approximately 60 pct for reproduction.

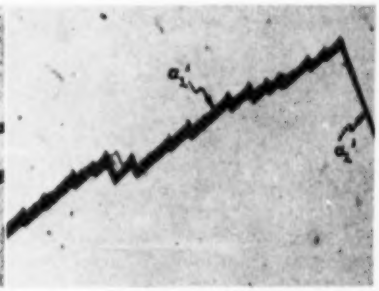

Fig. 11—Change in direction of propagation of martensite. X650. Area reduced approximately 60 pct for reproduction.

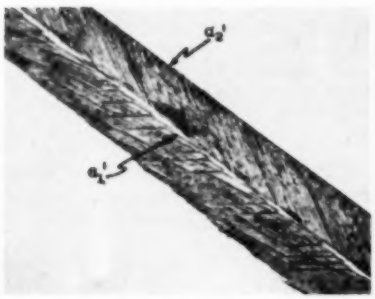

Fig. 12—Two parallel bands of martensite. X650. Area reduced approximately 60 pct for reproduction.

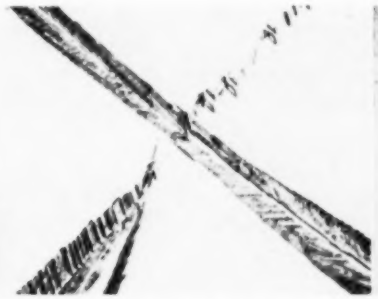

Fig. 13—Same two bands shown in Fig. 12 at point of separation. X650. Area reduced approximately 60 pct for reproduction.

Fig. 14—Parallel array of martensitic platelets. X650. Area reduced approximately 60 pct for reproduction.

Experimental Results and Discussion

Based on 22 measurements (two-surface) made on six grains in five crystals, the poles of the habit planes were found to fall distinctly into two groups, designated as α'_1 and α'_2 in Fig. 7, thus indicating the

Fig. 15—Another view of parallel martensitic platelets. X650. Area reduced approximately 60 pct for reproduction.

existence of two habit planes. The indices of these two habit planes were determined to be close to $\{334\}_\gamma$ and $\{344\}_\gamma$.

The martensite corresponding to the α'_1 habit occurs most frequently and appears as fine straight lines, Fig. 9. In more than 40 measurements on this type of structure, it was found that the trace normals, without exception, passed through the α'_1 region.

Figs. 10 and 11 show that the martensite platelet changes its direction of propagation quite readily. From 11 analyses, it has been found that the component parts of this type of structure always follow the α'_1 habit plane and that the poles of these two α'_1 planes always have an angular separation of 60° to 64° as measured on the stereographic projection.

A number of these interesting microstructures, in addition to those shown previously were also encountered during the present investigation. Some of these structures are shown in Figs. 9 to 15. Fig. 12 shows two parallel bands, one with smooth boundaries and the other jagged. These platelets generally are found together and the habit plane along which they occur is indicated. However, the submarkings within the bands cannot be accounted for on the basis of α'_1 or α'_2 planes of the matrix. This type of structure is the thickest observed, the estimated thickness being less than 0.001 in. Fig. 13 shows the same structure as Fig. 12. At the end of this coupled structure, the platelets tend to separate. The thickness of the platelets is reduced when they meet other martensitic transformation products.

Other types of substructure are often found in the martensite platelet. In one type, the substructure frequently extends into the untransformed matrix and in others it resembles twin formation in hexagonal close-packed metals. Fig. 9 shows dark-etching needles which are not confined to the main platelet, but other parallel markings which are confined are also visible. Fig. 10 shows "twins" within a martensite platelet.

The transformation structure is sometimes seen as a cluster of closely-separated parallel lines, as shown in Figs. 14 and 15. These markings can run either transverse or almost parallel to the direction of propagation.

In addition to the above, the following observations on the martensitic structure have also been made:

1—The mid-rib structure, characteristic of martensite structure in steel, does not occur in the Ti-Mn system.

2—Any boundary between the platelets and matrix corresponds, without exception, to one of the two habit planes found.

3—Submarkings within martensite cannot always be accounted for on the basis of α'_1 or α'_2.

Summary

1—Two habit planes of martensite with Miller indices close to $\{334\}_\gamma$ and $\{344\}_\gamma$ were determined in the Ti-Mn system.

2—A variety of martensite microstructures have been found in the present alloy system. Many of these are quite different from the martensitic structures observed in steel, and in none of the structures is a mid-rib observed.

Acknowledgments

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The authors also wish to express their appreciation to E. S. Machlin of Columbia University for his helpful discussions, and to L. D. Jaffe of Watertown Arsenal Laboratory for his interest and encouragement.

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Technical Note

Observations on the Phase TiAg

by R. J. Van Thyne, W. Rostoker, and H. D. Kessler

THE existence of the phase TiAg has not been reported previously. Other investigators have denied the existence of compounds in the Ti-Ag system.[1,2] However, phases of equi-atomic proportion have been reported by Raub, et al.[3] in the (similar) systems of titanium with gold and copper and zirconium with silver. Raub's work indicated that titanium and silver are immiscible in the liquid state. No evidence of this was found although a very wide melting range exists which makes the preparation of homogeneous alloys difficult. TiAg is apparently formed by a peritectic reaction between β

titanium and a silver-rich liquid phase.

The 68 pct Ag-Ti alloys were melted in a nonconsumable tungsten electrode arc-melting furnace. A detailed description of the melting techniques may be found elsewhere.[2] Two alloys were prepared. Iodide titanium and a very pure grade of silver were used in the preparation of one alloy and sponge titanium was used for the other alloy. Fig. 1 illustrates the single phase microstructure of the alloy prepared with iodide titanium, annealed at 900°C for 24 hr.

The hardness of TiAg is quite low for an intermediate phase. A hardness value of 130 diamond pyramid hardness was obtained for both the higher and lower purity alloys. The alloy prepared with sponge titanium was cold-pressed 75 pct although edge cracking resulted.

The structure of the TiAg phase was determined by X-ray diffraction analysis. Powder patterns were taken on a 14 cm diam camera using filtered Kα copper radiation. By trial it was found that the lines could be indexed on a tetragonal Hull-Davey type chart for a unit cell having an axial ratio just less than unity.

The relative intensities of the lines indicated a CuAu ordered face-centered tetragonal structure in which the atoms of one species are located at the (0, 0, 0) and ($\frac{1}{2}$, $\frac{1}{2}$, 0) positions and of the other species, at the (0, $\frac{1}{2}$, $\frac{1}{2}$) and ($\frac{1}{2}$, 0, $\frac{1}{2}$) positions in the unit cell. On the assumption that this was the correct structure, the relative intensities of the lines were calculated by means of the familiar equation

$$I_{hkl} \propto p \cdot F^2 \cdot \frac{1 + \cos^2 2\theta}{\sin^2\theta \cos\theta}$$

where p is the multiplicity factor; F, the structure factor; and θ, the Bragg angle. Actual relative line intensities were measured with aid of a Norelco type Geiger counter spectrometer.

The dimensions of the unit cell were computed by a least squares solution of equations for interplanar spacing of five high angle lines. The results were as follows: $a = 4.096$ kX, $c = 4.069$ kX, $c/a = 0.993$. Using these values, the interplanar spacings of each of the indexed lines were calculated and compared to the observed values. Table I summarizes the calculated and observed d values and relative intensities of the indexed lines.

Table I. Comparison of Calculated and Observed Values of Line Spacings (kX Units) and Relative Line Intensities

hkl	d (Observed)	d (Calculated)*	I (Observed)	I (Calculated)
110	2.728	2.896	15	16
111	2.357	2.360	100	100
200	2.050	2.048	29	35
002	2.030	2.039	16	18
112	1.634	1.665	5	6
220	1.449	1.448	10†	7
202	1.443	1.443	18	23
311	1.235	1.234	14	14
113	1.228	1.228	8	13
222	1.182	1.180	7	12
312	1.090	1.093	2	3
004	1.014	1.017	2	2
331	0.9380	0.9394	8	7
420	0.9166	0.9159	6	7
204	0.9117	0.9111	6	7

* Calculated from the least squares solution for the c and a parameters.

† Intensity is anomalously high because of superposition on (202) line.

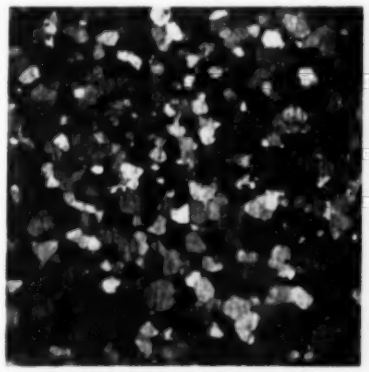

Fig. 1—68 pct Ag-Ti alloy annealed at 900°C for 24 hr and water-quenched. The intermediate phase, TiAg. Unetched, polarized light. X250. Area reduced approximately 50 pct for reproduction.

It is interesting to note that TiAg has a similar structure to TiAl[4-6] although the latter phase has an axial lattice ratio slightly greater than one. Both phases are soft and can be deformed cold by compression; however, TiAl is not as ductile as TiAg.

Acknowledgments

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TN 156E. Manuscript, Jan. 23, 1953.

Recrystallization of a Cold-Rolled Copper Single Crystal

by Y. C. Liu and W. R. Hibbard, Jr.

Based on pole figure data and microstructural observations, the recrystallization orientation found in a copper strip previously cold-rolled 99.5 pct from a single crystal with an initial (110) [112] orientation may be described as: a 30° rotation, clockwise and counterclockwise, about octahedral poles of the cold-rolled texture such that all four poles function at a low annealing temperature (400°C) and only three of them function at high temperatures (500° to 1050°C). The relative intensity of deformation stresses on various slip planes can be correlated with the choice of poles affecting the rotations found in the recrystallized orientations.

ROTATIONAL reorientation about the poles of densely packed crystallographic planes is an important characteristic of secondary recrystallization[1-4] in face-centered cubic metals. Some evidence has also been interpreted to show that such a rotational relationship also exists in the primary recrystallization process.[5] In an investigation of the crystallographic relationships in the recrystallization process, several aspects of the experimental procedure should be considered. With cold-rolled polycrystalline metal as the initial material, pole figure analysis leaves many ambiguities as to whether or not simple indices can be assigned to adequately represent its orientation. The rolling texture does not consist of a single texture. Minor deformation textures are usually present. In all cases, twin textures are present. The presence of both minor and twin deformation textures influences the recrystallization texture. In order to study the recrystallization mechanism, a more precise knowledge of the history of the material before recrystallization is necessary. Tensile deformation of a single crystal usually produces residual stresses which are concentrated at slip bands. Since the exact nature and orientation of these potential nuclei sites cannot be experimentally evaluated, the tensile deformation of a single crystal was not considered adequate.

From the work done by Barrett and Steadman[6] on copper, it appeared that cold rolling of a single-crystal specimen with an initial (110) [112] orientation would yield a specimen which fulfills the conditions required for the present investigation, namely a highly preferred single rolling texture with relatively homogeneous stress distribution. By investigating the recrystallization texture of this material, additional details of reorientation during recrystallization might be obtained. The purpose of this paper is to describe such an investigation.

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Discussion on this paper, TP 3527E, may be sent, 2 copies, to AIME by July 1, 1953. Manuscript, Sept. 10, 1952; revision, Feb. 10, 1953. Los Angeles Meeting, February 1953.

This paper represents part of a thesis by Y. C. Liu submitted in partial fulfillment of requirements for the degree of Doctor of Engineering to the Faculty of the Engineering School of Yale University, May 1952.

Experimental Procedure

Copper used in the present investigation was cathode sheet with a purity of 99.94 to 99.97 pct. A copper single-crystal specimen, 1.15x0.8720x0.5322 in. (thickness) was cut from a cylindrical crystal which was grown in a Bridgman furnace.[7] The cutting was accomplished on a horizontal milling machine operated at a very slow speed to obtain a (110) plane in the rolling plane and a [112] direction in the rolling direction. The disturbed surface was removed by electrolytic polishing in dilute orthophosphoric acid (H_3PO_4) with a specific gravity of 1.14, and a current density of about 1 to 2 amp per sq cm. A 3-min polishing was needed to eliminate the disturbed surface causing Debye-Scherrer rings and a 45-min polish produced a surface which yielded sharp rounded Laue spots. It was estimated that about 1/16 in. of metal was removed from the original cut surface. The final orientation of this specimen before rolling was about 2° from the (110) plane in the rolling plane with a [112] direction aligned in the rolling direction. It is possible that a small amount of strain was still present in the cube despite the fact that the Laue spots seemed very sharp. It is doubtful, however, that this would be an important factor after a subsequent rolling reduction of 99.5 pct.

This copper specimen was rolled on a laboratory rolling mill with two highly polished rolls 3⅞ in. in diam in the following manner: 0.010 in. per pass to 50 pct reduction, 0.005 in. to 70 pct, 0.002 in. to 85 pct and, finally, 0.001 in. per pass to a 99.5 pct reduction.

Specimens, about 1 in. sq with the rolling edges intact, were cut from the rolled strip by the electrolytic method previously described, after the strip

was covered with electroplater's masking tape, with the exception of a thin slit at the place where it was desired to cut the specimen.

Annealing was carried out in an electrically heated liquid metal or salt bath where possible. Each specimen was dusted with alumina powder and placed between two copper sheets 0.020 in. thick. The assembled sandwich was wrapped with several layers of copper foil 0.002 in. thick. A thermocouple was fastened to each envelope, so that the exact temperature of the envelope could be measured. In all cases, the temperature variation during annealing was controlled within ±5°C. In the annealing of the specimen at 1050°C in an air furnace, it was found that the furnace temperature dropped to 980°C upon the introduction of the specimen. A 30-min (1800-sec) reheating period was required prior to the 300 sec at 1050°C.

Cold-rolling textures were determined by ordinary pole figure construction[8, 9] with the help of the glancing X-ray method[10] to complete the center portion of the pole figure. An integrating camera[11] was employed in the construction of the recrystallization pole figures because of the large grain size. Since all the recrystallization pole figures were asymmetrical about either the rolling direction or transverse direction, the ordinary basic chart for the construction of pole figures was modified to cover all four quadrants.

Metallographic specimens were electrolytically polished and etched according to the ASTM method for copper and its alloys.[12] Tracings of microstructures were made after each heat treatment and the percentage of recrystallization was determined by averaging 15 measurements of the number of squares of cross-section graph paper encompassed.

Results and Discussion

A single-crystal specimen of copper, with a (110) plane parallel to the rolling plane, was rolled in a [112] direction lying in the rolling direction. The resulting pole figure, Fig. 1, shows that even after 99.5 pct reduction in thickness, the rolled strip still retained essentially its original orientation with about 8° of scatter. The cold-rolled texture has no twin component. This experiment confirms earlier work done under the similar conditions in copper[6] and in brass[13] and recently in a Cu-Si alloy.[14] Since the specimen, after 99.5 pct reduction, was less than 0.003 in. thick, it was not expected that there would be a distinct difference of cold-rolled texture on the specimen surface and in the interior of the specimen as had been reported.[15-16]

Table I. Recrystallization Measured in Annealed Specimens

Annealing Temperature, °C	Annealing Time	Percentage of Recrystallization
200	One week	Less than 1 pct
300	300 sec	None observed
300	1000 sec	First trace of recrystallization visible at 400X
400	100 sec	No observation
400	300 sec	67.8
400	1000 sec	68.0
500	30 sec	78.1
500	100 sec	88.2
500	300 sec	95 (estimated)
700	300 sec	100
900	300 sec	100
980-1050	1800 sec	100
1050	300 sec	

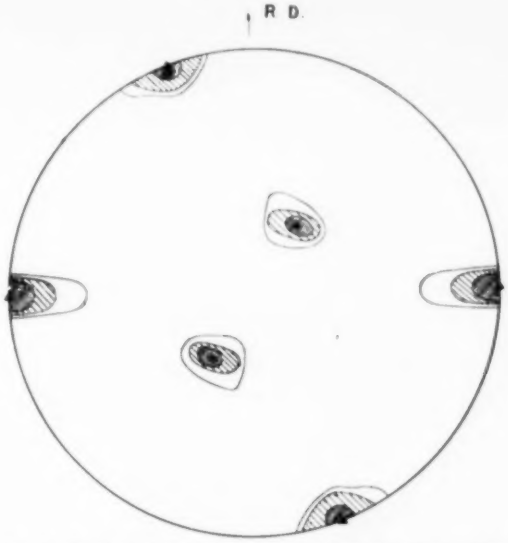

Fig 1—Octahedral pole figure of cold-rolled copper strip after 99.5 pct reduction in thickness.

A series of specimens from the same rolled strip was annealed at temperatures ranging from 200° to 1050°C. The percentage of recrystallization measured in the annealed specimens is listed in Table I. These data show that the range of annealing temperatures and times includes the entire recrystallization process.

The recrystallization texture of all annealed specimens was plotted in the form of conventional octahedral pole figures in Figs. 2 to 10. In general, successive pole figures are quite consistent within the experimental error and possess one center of symmetry, namely a 180° rotation about the center of the figure. Specimens annealed at lower temperatures yielded pole figures containing more high intensity areas than those annealed at higher temperatures. In the latter case, however, these areas were more compact and of somewhat higher intensity. The range and the complexity of the results make it impossible to describe the recrystallization textures adequately by means of simple Miller indices. The predominately cubic recrystallization texture of polycrystalline copper strip was not found in the present specimens.

By careful analysis of the pole figures, a relationship was found between the orientation of the recrystallized grains and the deformation texture. If the deformation orientation is rotated about each one of its octahedral poles, 30° both clockwise and counterclockwise, the resulting ideal orientations were found to best fit the high intensity areas of the pole figures of recrystallized specimens. The theoretical orientations obtained from the rotations described above are plotted in Fig. 11 and these orientations are also plotted as circles (both open and solid) in the individual pole figures in Figs. 4 to 10. The results fit the data very well, especially for the specimens annealed at lower temperatures. The 30° type rotation about the (111) type pole is similar to prior results on a rolled copper single crystal[17] recrystallized at 450°C, although the latter was not rolled in the critical (110) [112] orientation.

The selection of the 30° rotation, both clockwise and counterclockwise, was made on the basis of the fact that the resulting ideal orientations were then

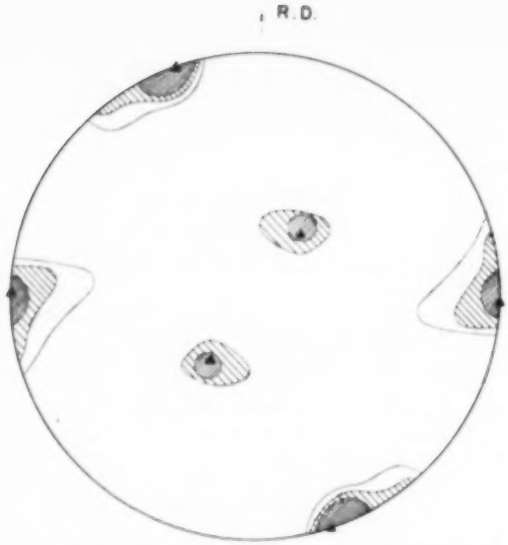

Fig. 2—Octahedral pole figure of copper specimen annealed one week at 200°C.

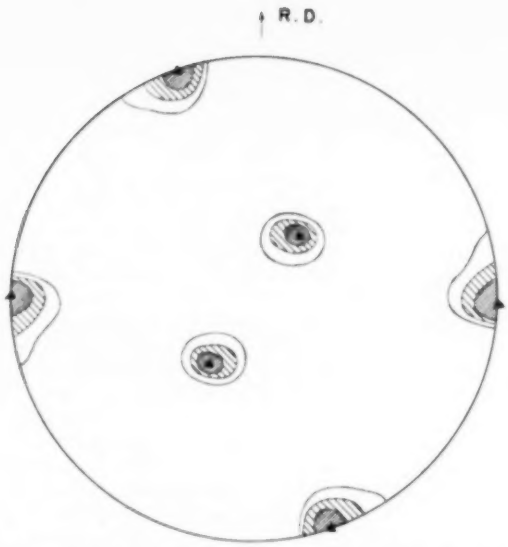

Fig. 3—Octahedral pole figure of copper specimen annealed 300 sec at 300°C.

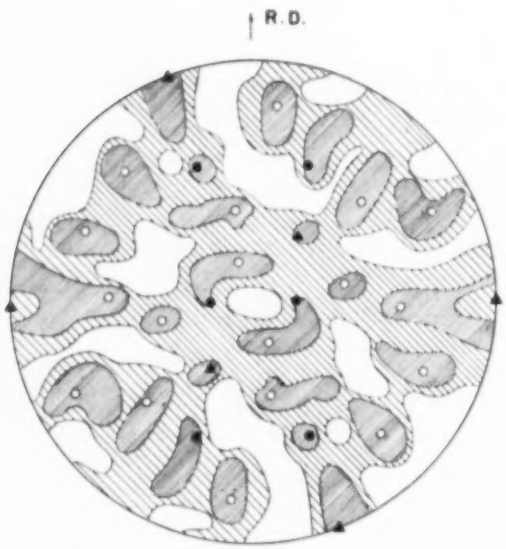

Fig. 4—Octahedral pole figure of copper specimen annealed 100 sec at 400°C.

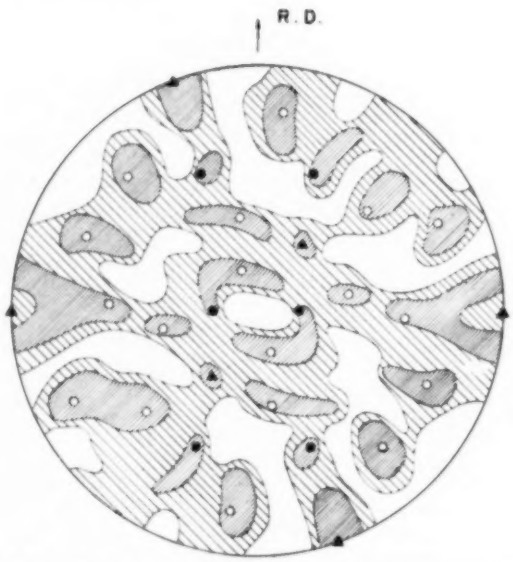

Fig. 5—Octahedral pole figure of copper specimen annealed 300 sec at 400°C.

located for the most part at the center of these intense areas. Actually, any combination of pairs of opposite rotations in the range of 26° to 34°, totaling 60° would be suitable. Opposite rotations totaling 60° about an octahedral pole will generate a twin orientation across the plane of this octahedral pole in face-centered cubic metals. It is possible that orientations, other than those resulting from the rotations suggested, will also fit the data, but it is doubtful that these orientations will satisfy all of the intense areas and also have a simple relationship among themselves.

For annealing temperature at and above 500°C, differences between the ideal orientations and the experimental data occur as shown in Figs. 7 to 10. Two relatively intense areas near the periphery at 1 and 7 o'clock have not been accounted for and six (solid) circles of the theoretical orientations resulting from rotations about octahedral pole IV, are not in intense areas. The failure of the recrystallized grains to include an orientation relationship about octahedral pole IV at higher annealing temperature might be interpreted as the result of either: 1—the recrystallized grains resulting from the rotational reorientation about pole IV being absorbed during grain growth, or 2—the rotational reorientation process not functioning with respect to octahedral pole IV.

The first alternative, that grains related to pole IV are actively generated at the start of recrystallization but are absorbed by other growing grains, requires that this absorption occur between 400°C (Fig. 6) and 500°C (Fig. 7) as indicated by the loss of intensity about the orientations with the solid circles. Such a selective absorption from the four equivalent groups of orientations is considered im-

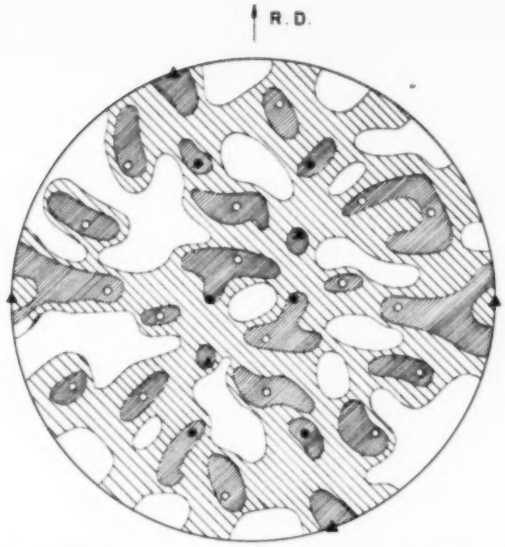

Fig. 6—Octahedral pole figure of copper specimen annealed 1000 sec at 400°C.

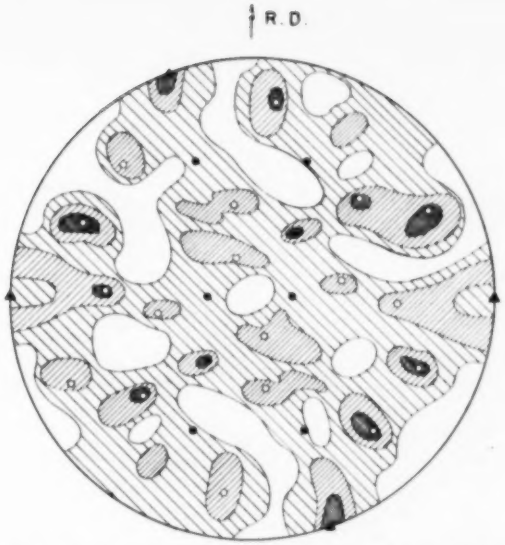

Fig. 7—Octahedral pole figure of copper specimen annealed 300 sec at 500°C.

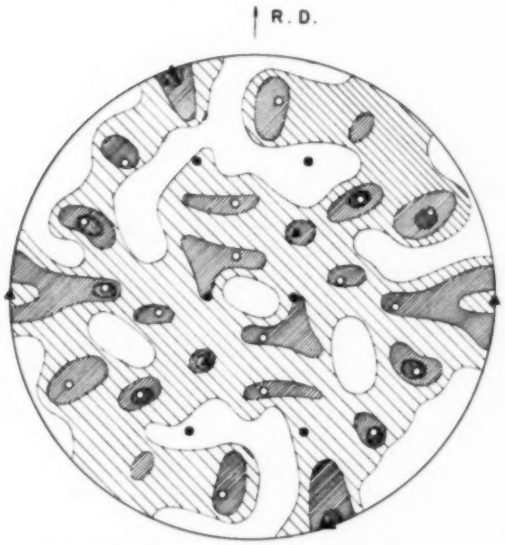

Fig. 8—Octahedral pole figure of copper specimen annealed 300 sec at 700°C.

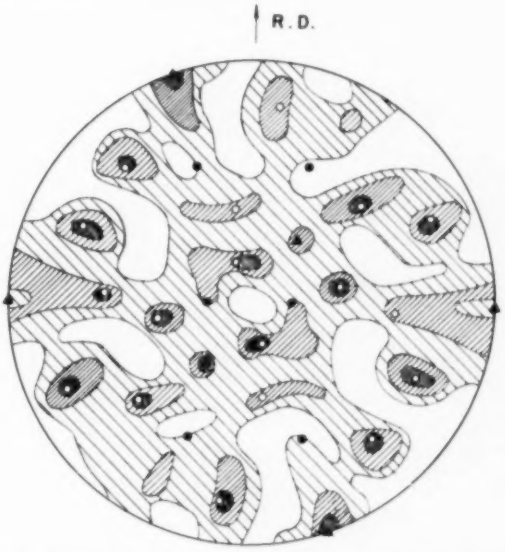

Fig. 9—Octahedral pole figure of copper specimen annealed 300 sec at 900°C.

probable. Therefore, it is suggested that the difference between the recrystallization textures for high and low temperatures is the activation of nuclei related to pole IV at low temperatures, but not at high temperatures.

The second possibility mentioned above may be considered in terms of deformation stresses on the octahedral slip planes based on single-crystal behavior during cold-rolling and correlated with subsequent recrystallization. Maddin, Mathewson, and Hibbard, working on brass,[18] and Chen and Mathewson, working on aluminum,[19] have reported that the orientations of recrystallized grains may be related to the orientation of the deformed crystals by rotations about poles of the active slip planes.

Fig. 12 shows the positions of [111] poles and [110] directions for the starting orientation, (110) [112]. The equation for resolved shear stress is written as

$$S = \frac{F}{A} \cos \phi \cos \lambda \qquad [1]$$

where ϕ is the angle between stress axis and normal of the slip plane and λ is the angle between stress axis and slip direction.

If slip takes place during cold rolling in accordance with the resolved shear stress considerations based on plane parallelopiped-compression, it is clear that the plane IV, which is perpendicular to the rolling plane and parallel to rolling direction, has a resolved shear stress equal to zero. Theoretically, this slip plane should not operate during cold rolling and at most it would be expected that this slip plane functions least in comparison with the other three octahedral planes. If the factor of stresses on an octahedral plane is assumed to influence recrystal-

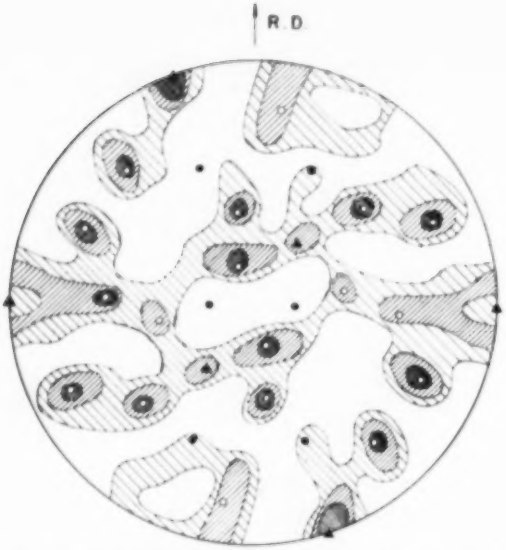

Fig. 10—Octahedral pole figure of copper specimen annealed 1800 sec at 980° to 1050°C and 300 sec at 1050°C.

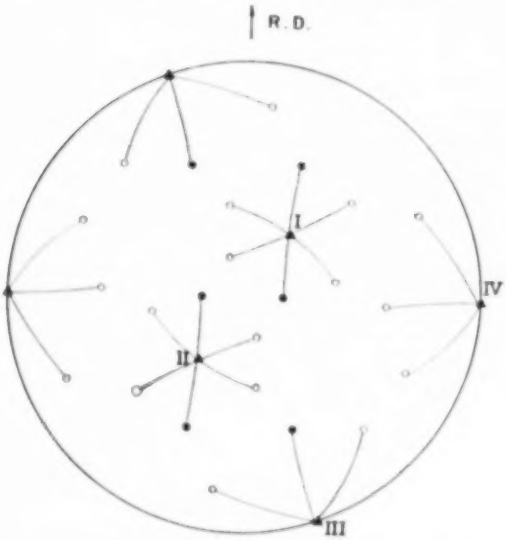

Fig. 11—Ideal orientations resulting from the proposed recrystallization rotations.

lization, then it follows that the low stresses on plane IV rendered the plane least potent in the recrystallization process. The stresses on plane IV, then, may be sufficiently large to cause plane IV to contribute to recrystallization below 500°C, but yet small enough so that they are relieved at temperatures above 400°C rapidly enough and to the extent that plane IV does not contribute to recrystallization at these temperatures.*

* This interpretation is consistent with the results of Maddin, Mathewson and Hibbard[18] and Chen and Mathewson[19] that, after annealing at high temperatures (800° and 600°C), new orientations could be related only to the poles of those planes which were active during deformation. These findings suggest a significant dependence of the activation of nuclei on stress configuration and a competitive relation between the rate of stress relief and the nucleation frequency at the annealing temperature. It has been reported that the nucleation frequency is dependent upon both the amount of strain and stress relief.[20-23] These factors also affect the incubation time for recrystallization, and therefore the time available for stress relief. Thus, it is more probable that stress relief can occur during the incubation period for potential nuclei in regions of small strain and low residual stress such as along plane IV, provided the temperature is sufficiently high.

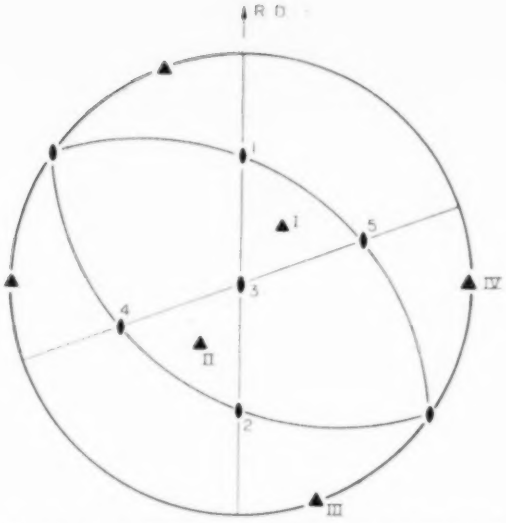

Fig. 12—The positions of octahedral poles and dodecahedral directions in a (110) [112] orientation.

Fig. 10 is the pole figure plotted for the specimen annealed 1800 sec at 980° to 1050°C and 300 sec at 1050°C. The area around [111] poles I and II are less intense than the area around [111] pole III. If this phenomenon is dependent on a stress factor, then it would be expected that the stresses are higher on plane III than on planes I and II.

Hibbard and Yen[22] suggested that the stable end orientation of cold-rolled polycrystalline strips of face-centered cubic metals is (110) [112]. The slip systems that operate in a grain rolled in this orientation are systems I-2 and II-1 in Fig. 12. The combination of slip on these two operative slip systems produces a "flow" phenomenon, such that a grain rolled in (110) [112] orientation can experience further cold rolling without changing its orientation.

The role which the (111) plane III plays during cold rolling is complex. According to theory,[22] owing to the unfavorable orientation of slip directions in

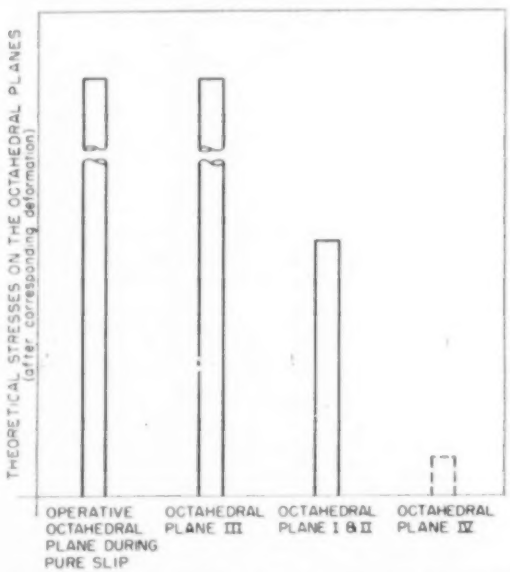
THEORETICAL STRESSES ON THE OCTAHEDRAL PLANES
(after corresponding deformation)
OPERATIVE OCTAHEDRAL PLANE DURING PURE SLIP
OCTAHEDRAL PLANE III
OCTAHEDRAL PLANE I & II
OCTAHEDRAL PLANE IV

Fig. 13—Schematic representation of comparative amount of residual stresses on the individual octahedral planes.

Fig. 14—Cold-rolled structure. Rolling direction is vertical. X500. Area reduced approximately 60 pct for reproduction.

Fig. 15—Specimen annealed 300 sec at 900°C. X500. Area reduced approximately 60 pct for reproduction.

Fig. 16—Specimen annealed 300 sec at 700°C. X500. Area reduced approximately 60 pct for reproduction.

Fig. 17—Specimen annealed 300 sec at 900°C. X500. Area reduced approximately 60 pct for reproduction.

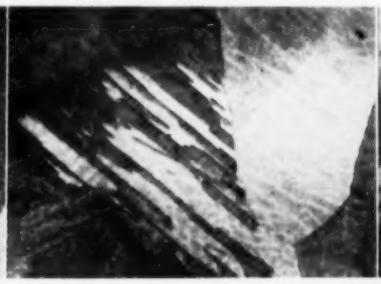

Fig. 18—Specimen annealed 300 sec at 700°C. X500. Area reduced approximately 60 pct for reproduction.

Fig. 19—Specimen annealed 1000 sec at 400°C. X500. Area reduced approximately 60 pct for reproduction.

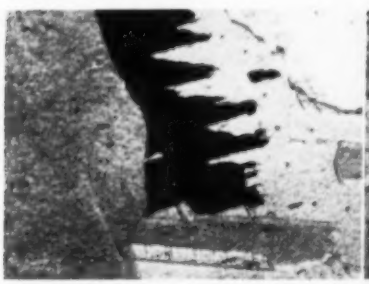

Fig. 20—Specimen annealed 300 sec at 500°C. X500. Area reduced approximately 60 pct for reproduction.

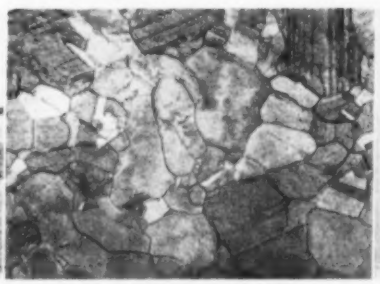

Fig. 21—Specimen annealed 300 sec at 900°C. X500. Area reduced approximately 60 pct for reproduction.

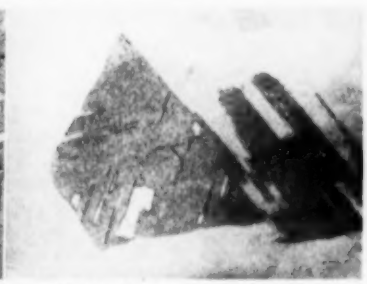

Fig. 22—Specimen annealed 300 sec at 700°C. Rolling direction is vertical. X500. Area reduced approximately 60 pct for reproduction.

this plane, III, with respect to the rolling direction, this slip plane should not operate during cold rolling without a change in orientation. The location of the slip plane causes the cos ϕ value in Eq. 1 to become large with respect to the tensile component in the rolling direction due to cold rolling. In that way, the plane itself would be subjected to large stresses built up during deformation, but the unfavorable position of the slip directions with respect to flow restricts slip. Under such conditions a high stress is built up on plane III. Slip on planes I and II may relieve a portion of this stress, but that remaining on plane III is probably higher than that on planes I and II subsequent to their yielding. The role of slip plane III in this situation might be analogous to that of the cross slip plane in tensile specimens. As reported by Maddin, Mathewson, and Hibbard in brass[14] and Chen and Mathewson in aluminum single crystals,[15] the subsequent anneal of those specimens yielded the recrystallized grains which were more frequently related crystallographically to the poles of cross slip planes than to the poles of primary and conjugate slip planes.

A schematic representation of the comparative amount of stress on the individual [111] planes of the deformation texture is shown in Fig. 13. The relatively high stresses existing on the [111] plane, III, is believed to be responsible for the observation that the most intense areas of the pole figures of recrystallized specimens may be related to rotations about the pole of this plane III.

The presence of intense areas around the solid triangles in Figs. 8 and 9 may lead to the conclusion that some of the recrystallized grains may have an orientation similar to the deformed matrix, since, at these annealing temperatures, it has been found that these specimens have experienced full recrystallization. The analysis of the pole figure intensities at the four [111] poles of the deformed structure (solid triangles) is not a true way of estimating the number of recrystallized grains with this orientation. Since each of these four poles acts individually, in a crystallographic sense, as a center of rotation to develop the recrystallized orientations, these four [111] poles are therefore each associated with other orientations of newly formed grains and are not necessarily combined into a single orientation. It is not known whether (110) [112] type grains, if they

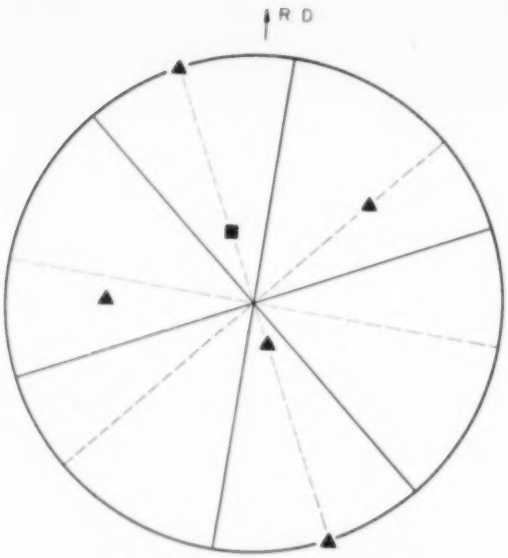

Fig. 23—Stereographic analysis of Fig. 22.

exist, are the result of the rotational reorientation or "recrystallization in situ."

As indicated by Table I, the temperature range used in this experiment covers the range of the recrystallization process. By comparing Figs. 4, 5, and 6, it is clearly seen that no significant change in the recrystallization texture has been produced by time during isothermal annealing. Also, in general the annealing temperature has a rather small effect on the orientation of the recrystallized grains, provided annealing has been carried out above 500°C.

Microscopically, the recrystallization process in the present specimen is entirely different from what has been reported previously for the polycrystalline copper strip. The specimen was very reluctant to recrystallize. No sign of recrystallization was observed after an anneal of 300 sec at 300°C. On increasing the annealing time to 1000 sec some recrystallization was observed. Annealed for a week at 200°C, the amount of recrystallization was estimated as less than 1 pct.

At a very early stage of recrystallization, a few large grains were found together with a small number of fine grains. In no case was a grain of circular outline observed. No attempt was made to measure the rates of nucleation and growth in the present investigation.

The characteristic feature of the microstructure of the recrystallized grains in the present specimens is the abundance of annealing twins. Figs. 14 to 18 show the typical structure of these specimens. It is interesting that the textures are not a twinned type in the usual sense (i.e., not twinned across planes perpendicular to the rolling plane and containing the rolling direction or the transverse direction), although they possess two-fold rotational symmetry.

Preferential growth of the recrystallized grains takes place during the recrystallization process. Figs. 19 and 20 show that the grain boundaries of the recrystallized grains are very jagged and extend finger-like into the deformed matrix. Even the microstructure of the specimen which has been annealed 300 sec at 900°C, Fig. 21, shows grain boundaries which clearly indicate that the equilibrium shape of grains has not yet been attained.

In testing the validity of the results obtained on the basis of the pole figure data, the orientation of certain grains could be determined by means of three twin families. It was expected that these orientations so independently determined should be among the eight orientations produced by the recrystallization rotations described above. All the measurements made both on the micrographs and under metallographic inspection conformed to this expectation. One of the micrographs, Fig. 22, is, therefore, reproduced here with the stereographic analysis.†

† In the stereographic analysis, the solid lines represent the traces of the twins in the corresponding micrograph, and the dotted lines represent the locus of octahedral poles corresponding in turn to these solid lines. The solid triangles represent the [111] poles of one of the eight equivalent orientations suggested by the recrystallization mechanism.

From both the pole figure and microstructure studies, it may be concluded that the recrystallization reorientations suggested in the present investigation are in good agreement with experimental observations.

Summary

1—A single-crystal specimen of cathode copper was cut with the (110) plane parallel to the rolling plane and was rolled in a [112] direction. After cold rolling, a pole figure showed that the final strip retained its original orientation with a spread of about 8° after a 99.5 pct reduction in thickness.

2—Specimens electrolytically cut from the cold-rolled strip were annealed at temperatures and times ranging from the initial stages of recrystallization to just below the melting point of copper.

3—Qualitatively, increasing the annealing temperature causes little change in the final recrystallization texture, provided the annealing temperature is above 500°C in the present specimen. Annealing time had no measurable effect on the orientations of the recrystallized grains.

4—The recrystallization reorientation found in the present investigation can be described as: 30° rotation clockwise and counterclockwise, about each of four [111] poles of the deformation texture at low annealing temperatures (up to 400°C) and only three at higher annealing temperatures (500°C and above).

5—The distribution of deformation stresses is considered an important factor which influences recrystallization reorientations.

Acknowledgment

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Technical Note

On the Distribution of Sodium in Modified Al-Si Alloys

by A. B. Michael and M. B. Bever

THE structure of cast Al-Si alloys is altered profoundly by modifying agents. Sodium, in particular, reduces the size of the silicon particles in the eutectic and tends to change their shapes from plates to nearly rounded grains. This refinement is accompanied by a shift of the eutectic to higher silicon concentrations and lower temperatures. Similar effects can be attained by rapid solidification.

Hypotheses advanced to explain modification have involved the formation of a ternary eutectic,[1] the removal of oxides by fluxing,[2] and even a presumed allotropic transformation of silicon. Other hypotheses assume that the modifying agent causes an envelopment of the silicon particles by the aluminum-rich solid solution, but attribute this envelopment to different mechanisms. According to various explanations, the growth of the silicon particles is obstructed by an immiscible sodium-rich liquid,[3] an adsorbed layer of sodium,[4] or a ternary compound;[5] this obstruction has been conceived as purely mechanical or as a colloidal action.[6] It has also been proposed that sodium reduces the surface tension on the Al-Si interface, thus permitting the aluminum-rich solid to grow faster and to envelop the silicon particles.[7] Still other hypotheses stress undercooling and the attendant change in the rate of nucleation of silicon, but fail to ascribe any definite function to the sodium addition.[8-10]

Procedure

High-purity alloys of aluminum and 3 or 7 pct Si were treated with modifying agents containing radioactive sodium tracers. The experiments were intended to investigate the partition of sodium between the primary aluminum-rich solution and the eutectic, and, if possible, to determine whether sodium is concentrated in or around the silicon particles in the eutectic. Radioactive sodium fluoride, alone or mixed with stable sodium chloride, was allowed to react with the alloy for 5 min immediately before solidification in the crucible. Modification could not be attained with additions of less than the equivalent of approximately 0.1 pct Na. Some alloys were not completely modified, but comparison of the structures of treated and untreated specimens indicated a very considerable reduction in particle size of the silicon in all of the former. A treated alloy is shown in Fig. 1.

Radioactive sodium fluoride was prepared by bombardment with deuterons having an energy of 14 mev. This bombardment (totaling, for example, 6 microampere-hours in one run) produced radioactive Na^{24} (half-life 14.8 hr) by the reaction $_{11}Na^{23}$ (d, p) $_{11}Na^{24}$. The radioisotopes of fluorine also produced have short half-lives and were not detected. A half-life determination by counting confirmed that the measured activity was due to Na^{24}.

The distribution of the sodium in the solidified alloys was determined by an autoradiographic technique by which a suitable stripping film is attached to a very thin metallographic specimen. After exposure and development, this emulsion remains superimposed on the specimen and may be examined microscopically or photographed by reflected light. The emulsion of the autoradiograph can be correlated directly with the underlying surface by simple adjustment in focus. The details of this technique have been described elsewhere.[11]

Results

Fig. 2 is a micrograph of the autoradiographic emulsion ("autoradiograph") lying directly above the area of eutectic in Fig. 1. Fig. 3 shows a similar autoradiograph of a region consisting of primary solid solution. The small black dots (tracks) in these autoradiographs are developed grains in the emul-

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TN 157E. Manuscript, Dec. 29, 1952.

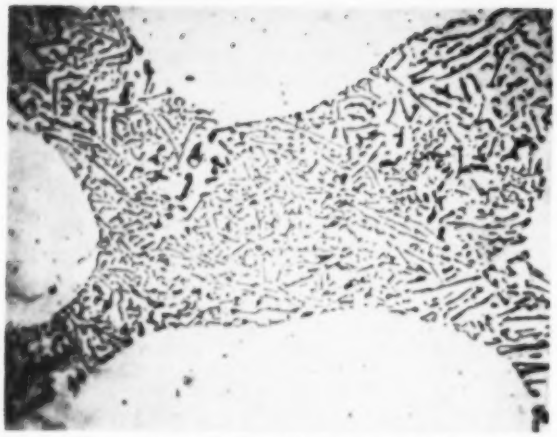

Fig. 1—Micrograph of metallographically prepared surface of alloy of aluminum and 7 pct Si, treated with modifying agent containing radioactive sodium. Etched with Keller's reagent. Photographed through autoradiographic emulsion. X500. Area reduced approximately 50 pct for reproduction.

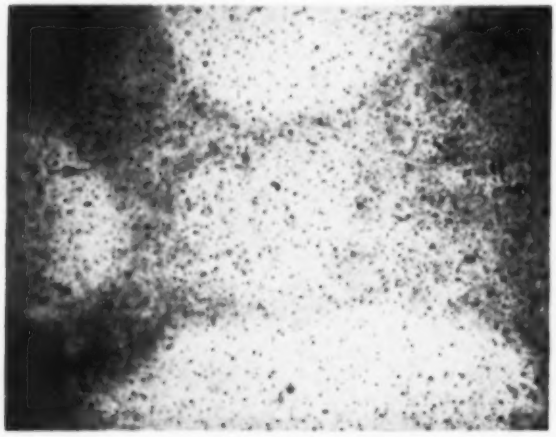

Fig. 2—Micrograph of autoradiograph corresponding to the area shown in Fig. 1. X500. Area reduced approximately 50 pct for reproduction.

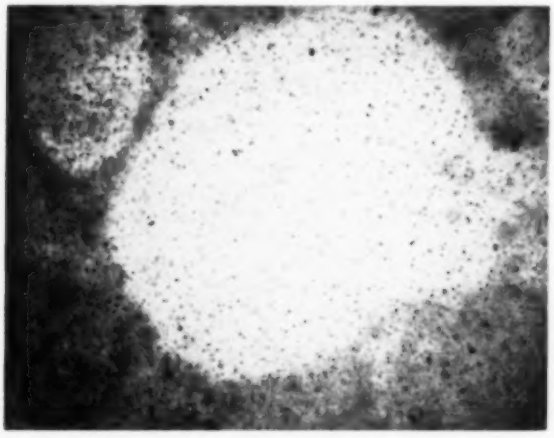

Fig. 3—Micrograph of autoradiograph of primary aluminum-rich solution. The area shown is adjacent to the area of Figs. 1 and 2. X500. Area reduced approximately 50 pct for reproduction.

sion due to radiations emitted by disintegrating sodium atoms. The uniform dark areas are eutectic in the underlying metal surface, which was not in focus when the micrographs of the emulsions were made. Blanks were processed simultaneously with the radioactive specimens. Their autoradiographs showed a negligible number of tracks.

The β tracks in the autoradiographs could not be traced to either of the two phases in the eutectic. The partition of sodium between the primary constituent and the eutectic, however, was determined by counting the number of developed grains in the emulsion above unit areas of these structures. These counts were made on ground glass images and on micrographs focused at a constant depth below the surface of the emulsion to eliminate possible variations with depth in the number of developed grains. Only the centers of large areas of eutectic or primary constituent were examined to avoid any effects due to cross radiation, which would occur near the boundaries between these structures. Since the thickness of the specimens was only about 20 microns, interference from subsurface layers was not appreciable.

In counts made on several specimens of the 3 and 7 pct Si alloys the number of developed grains was found to be approximately equal above regions of eutectic and of primary solid solution. This observation shows that sodium was not concentrated preferentially in the eutectic. It can be inferred as probable that in the eutectic sodium was associated with the aluminum-rich solid solution rather than with the silicon particles. In that case, the ratio of sodium to aluminum was higher for the eutectic than for the primary constituent, but this may be attributed to the lower temperature at which the eutectic solidifies from the residual liquid.

The finding that sodium was approximately equally distributed between the eutectic and the primary constituent and the inference that it probably was associated with the secondary aluminum-rich solid solution in the eutectic does not directly lead to a choice between the various hypotheses of modification. In particular, it can be argued that only a small fraction of the total sodium present may be effective in inducing modification and that the location and action of this effective portion is still not known. On the other hand, the observations reported here invalidate any hypothesis of modification which implies a close association of the bulk of the sodium with the silicon.

Support of this work through a fellowship sponsored by the Aluminum Company of America is gratefully acknowledged.

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AIME OFFICERS:

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Vice-Presidents—L. E. Elkins, O. B. J. Fraser, J. L. Gillson, A. B. Kinzel, J. B. Morrow
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Ass't. Treasurer—P. J. Apol

aime NEWS

Iron and Steel Dinner To Be Held on May 27

The annual May dinner of the Iron and Steel Div. of AIME will be held, as usual, on the night before the AISI banquet, May 27, 1953, at 7 pm at the Engineers' Club, 32 W. 40th St., New York City. Chairman T. J.

H. N. ALYEA

McLoughlin has obtained Professor Hubert N. Alyea of the Frick Chemical Laboratory, Princeton University, as speaker. His subject will be *Atomic Energy: Weapon for Peace.*

After graduating from Princeton University, Dr. Alyea spent several years doing post-graduate work, including two years in Sweden and Germany. He has done research work for the Office of Scientific Research and Development and taught at the University of Hawaii and Princeton.

In his talk, Dr. Alyea will trace the growth of ideas which lead to the atomic bomb and the tapping of nuclear energy. He will contrast the actions of ordinary high explosives with those of nuclear fission reactions. He will discuss the atomic pile, the production of new elements in it, research with radioactive isotopes, the use of the atomic pile as a source of energy and the nature of the H-fusion bomb. The lecture will be illustrated with chemical experiments and demonstrations.

All AIME members and guests are welcome. The price of the dinner will be $6 and cocktails will be available on the 10th floor of the Engineers' Club on an a la carte basis from 6 pm. Return postcards on which to make reservations are being sent this year only to those who have attended recent May dinners or who are members of Iron and Steel Div. Committees. If you did not get a card for this stag dinner, write E. O. Kirkendall, Secretary, Iron and Steel Div., AIME, 29 West 39th St., New York 18, N. Y., indicating the number of reservations desired.

Institute to Continue Student Prize Contest

Enough money has been raised to assure continuance of the National Student Prize Paper Contest on its present basis for some 13 years in the future. At the February Board meeting, held on February 15 in Los Angeles, Donald H. McLaughlin, chairman of a special committee to raise funds, reported that $6725 had been collected from 26 individuals and 9 corporations. Five prizes of $100 each are given annually, selection being made largely from those who have won Local Section prize contests. The donors of the $6725 were: F. M. Adams, Anaconda Copper Mining Co., A. H. Bunker, M. D. Cooper, E. DeGolyer, Cleveland Dodge, H. N. Eavenson, A. Fletcher, L. Gates, C. W. Hamilton, H. K. Hochschild, R. P. Koenig, J. A. Martino, D. H. McLaughlin, A. J. McNab, Paul D. Merica, Harvey S. Mudd, F. S. Mulock, M. Muskat, E. L. Oliver, R. G. Page, H. Shark, E. C. Smith, H. DeWitt Smith, Heath Steele, J. R. Suman, J. F. Thompson, American Smelting & Refining Co., Ingersoll-Rand Co., Inland Steel Co., International Minerals & Chemical Corp., Kennecott Copper Corp., New Jersey Zinc Co., U. S. Potash Co., and Wah Chang Corp.

Aluminum Monograph To Be Issued in June

First of a new monograph series resulting from world-wide research into the effects of adding various metals to iron and steel will be released in June of this year, by Alloys of Iron Research of the Engineering Foundation.

Aluminum in Iron and Steel, by S. L. Case and K. R. Van Horn will appear under the imprint of John Wiley & Sons, Inc. It contains a summary of data showing favorable effects of small amounts of aluminum when added as a deoxidizer to molten steel, and then correlates the data of the effects of aluminum as an alloying agent.

The monograph series will provide metallurgists, engineers, and other researchers with concise and comprehensive summaries of foreign and domestic data on the effects of selected metals on carbon steel, simple and complex alloy steels, and cast iron. Four other volumes will be published about four months apart, and are in various stages of preparation. Effects of nickel, manganese, titanium, columbium, zirconium, boron, tantalum, and calcium on iron and steel will be covered.

Centennial Pageant Is Recorded on Film

The pageant, "Adam to Atom," which thousands witnessed at the Museum of Science and Industry during the Centennial of Engineering at Chicago last September, was subsequently recorded on motion picture film. Prints are available from Ideal Pictures Corp., 233 W. 42d St., New York, N. Y. Distribution will be free to schools, luncheon groups, technical societies, and to any other responsible groups, by application to Ideal Pictures.

ANNUAL MEETING PAPER DEADLINE

Sept. 15, 1953 is the deadline for all Institute of Metals Div. papers for the 1954 New York Annual Meeting and the deadline for all Iron and Steel Div., and Extractive Metallurgy Div. papers to be prepublished.

RESEARCH IN PROGRESS

Research personnel are reminded of the Institute of Metals Div. Research in Progress session planned for during the Fall Meeting in Cleveland. Abstracts of proposed papers not exceeding 200 words in length are to be submitted to Institute headquarters prior to Sept. 1, 1953, for inclusion in the September issue of JOURNAL OF METALS. Abstracts not received in time for publication in the September issue may, nevertheless, be presented if received prior to Sept. 15, 1953, according to B. L. Averbach, IMD Program Committee.
The Program Committee urges those interested to avail themselves of the opportunity of participation in this program in order that new research information be presented for discussion.

E. Hassan Addresses San Francisco Section

While the U. S. is not a have not nation in respect to bauxite, present domestic reserves are limited with an unusually poor chance of being increased substantially by new discoveries, according to E. A. Hassan, Jr. of Kaiser Aluminum & Chemical Corp.

Mr. Hassan made the statement during an address before the San Francisco Section on *Jamaica—A New Source of Aluminum Ore*. He reviewed the history of discovery of Jamaican bauxite. Mr. Hassan noted that some 400 million tons of bauxite has been blocked out on the island.

He added that characteristics of the Caribbean ores markedly differ in character and mode of occurrence from those used to date in the western hemisphere. In both respects they resemble European Terra-rossa type of bauxite.

The Section recently held its formal dinner-dance in conjunction with the Ladies Auxiliary. The affair was held in Concert Room of the Palace Hotel.

Officers Elected By Connecticut Section

Members of the Connecticut Section unanimously elected the slate of officers presented by the nominating committee at the group's final technical meeting of the year held at the Connecticut Light & Power Co., auditorium, Waterbury, Conn.

Leon W. Thelin, Chase Brass & Copper Co., was named Chairman; Richard J. Christine, American Brass Co., Vice-Chairman; William E. Milligan, Yale University, Secretary-Treasurer; Delmar E. Trout, Scovill Mfg. Co., Chairman of Publicity; William L. Stearns, Bridgeport Brass Co., Chairman of Membership, and Albert I. Blank, Chase Brass & Copper Co., Member at Large.

Seventy members attended what was also the Section's annual meeting. Three technical papers were presented on the subject of the meeting, *Brass Mill Seminar*. Richard Russel, Chase Brass, spoke on *Piercing of Copper Alloys*. He reviewed the difficulty and solution of two problems in hot piercing of tube shells.

D. E. Trout addressed members on *Stress Relief Annealing of Sunk Brass Tubes*. He reviewed research on development of stress-relief heat treatments of hard drawn seamless brass tubes in roller hearth type furnaces. Frank Chudy, Bridgeport Brass, presented the third paper, *Some Adhesion Problems in Coiled Strip*. Two causes for adhesion were given. One was the effect of handling marks or dents, and the other tightness of the wound coil causing a welding of coils during annealing.

Only event remaining on the Section calendar for the year is the annual dinner dance May 15 at the Copper Kettle in Waterbury.

Principal of Douglas Fund Medal Increased

Through the good offices of Louis S. Cates, chairman of the Phelps Dodge Corp., the directors of that company have voted to give the AIME $11,800 to bring the principal of the James Douglas Medal Fund up to approximately $15,000. This amount is required to make possible an annual award of the gold medal. The James Douglas Medal was established in 1922, the first of the Institute's major medals, with the intention to bestow the award annually. Income in recent years has been insufficient, however; the last award was to Francis C. Frary in 1950.

The James Douglas Medal for distinguished achievement in nonferrous metallurgy was named in honor of Dr. Douglas, who twice served as President of the AIME, in 1899 and 1900; was Vice-President from 1906 to 1911; and elected an Honorary Member in 1906. Dr. Douglas was the first president of Phelps Dodge after its incorporation and long a dominant figure in the company. Though not educated as such he was a most ingenious, even an inspired, metallurgist. His benefactions were many, including $100,000 to the AIME to set up the James Douglas Library Fund.

Institute Fall Meetings Discontinued After 1953

In view of the continuing increase in the importance, number, and variety of Divisional and Branch fall meetings, it has been decided to abandon the all-Institute Fall Meetings after the meeting to be held in El Paso this year on October 29 to 31. Special circumstances may, however, call for exceptions to this general rule at some time in the future. Local Sections wishing to act as hosts to a Branch or Divisional Meeting should approach the Chairman of the Branch Council or Division whose group is most likely to be interested in coming to their area.

New Member Dues Now Prorated Monthly

The Council of Section Delegates has recommended that consideration be given by the Board to billing new members on a pro-rata basis for the number of months remaining in the calendar year of election. The Board later approved this recommendation and the text of the necessary change in the bylaws to accomplish this was published in the Institute journals of April 1952. The plan was not immediately adopted because further study was required at Institute headquarters. This having been completed, and the practicality of the plan assured, the bylaw revision was voted by the Executive Committee. The pertinent text of Art. II, Sec. 2, 3rd sentence (see p. xxxviii of the current Directory) is revised to read as follows: "Newly elected Members, Associate Members, or Junior Members shall, on receipt of Treasurer's statement, pay the current year's dues on a pro-rata basis for the months remaining in the calendar year, including the month of elections."

Henceforth, therefore, one month will be as good as another to join the AIME. If elected in June, for example, he would be billed for seven twelfths of the annual dues, and would receive the journal of his choice for the rest of the year.

I. E. Campbell Speaks At Philadelphia Section

I. E. Campbell of Battelle Memorial Institute spoke on *Vapor Deposition Processes* at the March meeting of the Philadelphia Section, held at the University of Pennsylvania.

Mr. Campbell outlined the various types of reactions used in the deposition of refractory metals, alloys, carbides, nitrides, borides, and silicides. Properties, applications, and tests of coatings were discussed.

Personals

A. J. SHALER

Amos J. Shaler has been appointed professor of metallurgy and chief of the division of metallurgy at Pennsylvania State College. Dr. Shaler was born in London and attended schools in Europe. He attended Massachusetts Institute of Technology, receiving his B.S. in 1940. He served in various capacities in the metallurgical industry and returned to MIT in 1945 to receive the degree of doctor of science. He was appointed assistant professor at MIT and in 1950 was named to serve as scientific liaison officer with the ONR in Europe. In July 1951, Prof. Shaler was named associate professor at MIT, a position he held until his recent appointment. An AIME Member, Dr. Shaler was awarded Rossiter W. Raymond Award in 1951 and has been active on the Powder Metallurgy Committee. In 1939 he was awarded the Singer-Polignac Medal. He holds membership in several professional societies and is vice-chairman of the board of *Acta Metallurgica*.

W. F. Huntley was named superintendent of the blast furnace dept., Jones & Laughlin Steel Corp., Aliquippa works. He was formerly assistant superintendent of the dept. Mr. Huntley succeeds **Elmer H. Riddle** who has retired.

John A. Deitrich has been appointed president of Hind Steel Co., Inc., Union, N. J. Prior to his appointment, Mr. Deitrich was divisional general manager of the Carpenter Steel Co.

James G. Hess, Jr., is with the Titanium Metals Corp. of America, Henderson, Nev.

Richard R. Snow, American Steel & Wire div., U. S. Steel Corp., has been named assistant to vice-president of operations, Cleveland. **John P. Debri** has been appointed general superintendent of the Wire div.'s Joliet, Ill. plant.

William E. Mahin recently resigned from the Armour Research Foundation, Chicago, where he had been director of research.

S. C. Faddis, former works metallurgist at the Pittsburgh works of the Jones & Laughlin Steel Corp., has been named assistant works manager of the Cleveland works div., succeeding **J. M. McColloch** who has retired.

James L. Clifford has been named district manager of the Udylite Corp.'s New York sales office.

Roland S. Young has resigned from the post of director of the Diamond Research Laboratory, Johannesburg, South Africa, to accept a position with the International Nickel Co. of Canada. Dr. Young will be attached to the New York office, but first will spend some time at the Clydack works of the Mond Nickel Co.

G. A. LILLIEQVIST

G. A. Lillieqvist, research director and chief metallurgist, American Steel Foundries, East Chicago, Ind. was awarded the Technical and Operating Gold Medal for 1952 by the Steel Founders' Society of America.

M. P. Drummond has been appointed manager of coke sales and **F. P. Jasper,** manager of chemical sales for the Colorado Fuel & Iron Corp.

William O. Harms was recently appointed to the staff of the Oak Ridge National Laboratory, Oak Ridge, Tenn.

August B. Hoefer has been elected vice-president of Udylite Corp. and vice-president and general manager of Frederic B. Stevens, Inc. The latter named firm is a subsidiary of Udylite Corp.

C. B. CARPENTER

Clark B. Carpenter, head of the metallurgy dept. and dean of the graduate school at the Colorado School of Mines, Golden, will retire this summer. Prof. Carpenter has served on the faculty since 1920 and been a full professor since 1936. He received his B.Sc. in mining from the University of Kansas in 1951 and an M.Sc. in mining from Massachusetts Institute of Technology in 1922. Prior to his joining the staff at Golden, he was associated with the Anaconda Copper Mining Co. and the Colorado Fuel & Iron Corp. For two months last year, Prof. Carpenter was guest lecturer at the Royal School of Mines, London, England. He will be accorded the rank of professor emeritus after his retirement. A Member of AIME, Prof. Carpenter is a Past Chairman of the Student Prize Paper Award Committee, was a member of the Student Relations Committee, and is also a Past Chairman of the Colorado Local Section. He has been a member of the Executive Committee of the Mineral Industry Education Div. and was very active in student relations work. Prof. Carpenter holds membership in several other professional societies and has written articles for various publications.

Ellis J. O'Brien has been appointed chief development engineer of Heyl & Patterson, Inc., Pittsburgh. Mr. O'Brien had been associated with the Vanadium Corp. of America as a process metallurgist.

W. J. Rundle has returned to the University of Wisconsin, after a year's leave of absence to work for the government in Washington.

Harold E. Rowen, vice-president and general manager of Sintering Machinery Corp., Netcong, N. J., will also direct sales of the company's metallurgical div.

R. B. SOSMAN

Robert B. Sosman, professor of ceramics, Rutgers University, was awarded the Albert Victor Bleininger Memorial Award by the Pittsburgh Section of the American Ceramic Society. The award is for distinguished achievement in the field of ceramics.

K. B. Stuart has been appointed general manager of coke and chemical sales for the Colorado Fuel & Iron Corp.

Robert D. Bell is presently associated with the Cerro de Pasco Corp., Oroya, Peru.

Max M. Muller and **Harvey N. Barrett, Jr.** have been elected vice-presidents of Basic Refractories, Inc., Cleveland. Dr. Muller will direct operations and engineering and Mr. Barrett will take charge of sales activities.

John A. Hipple, chief of the atomic section, National Bureau of Standard, Washington, D. C., has been named director of the Mineral Industries Experiment Station, Pennsylvania State College.

Dusan Pavlovic is metallurgical engineer with the Westinghouse Electric Corp., materials engineering dept., East Pittsburgh, Pa.

Richard R. Knobler will be temporarily located in Lima, Peru as chief metallurgist for M. Hochschild interests in South America.

NECROLOGY

Date Elected	Name	Date of Death
1928	Clifford P. Bowie	Apr. 1, 1952
1942	Sanford H. Casteel	June 20, 1952
1903	Firmin V. Desloge	May 18, 1952
1948	H. T. Herivel	Nov. 17, 1952
1905	Albert W. Johnston	Unknown
1944	Albert J. Jones	Dec. 1, 1952
1907	Charles Lessner	Unknown
1918	Alexandre Lheraud	December 1951
1941	Poole Maynard	Aug. 22, 1952
1948	Frank G. McClure	Mar. 2, 1953
1938	David M. Miller	Mar. 8, 1952
1884	Philip S. Morse	Mar. 1, 1953
1946	Walter J. Mullally	January 1952
1927	J. S. Peterson	Mar. 13, 1953
1935	Earl G. Schulz	Unknown
1886	Edward A. Uehling	Dec. 21, 1952

Obituaries

John Holger Gillies (Member 1951) died on Jan. 21, 1953. Born in Norway, Mr. Gillies was educated at the University of Glasgow, Scotland and Temple University. In 1934 he joined the Scott Paper Co., Chester, Pa. as a mechanical engineer. He joined the Canadian Navy in 1941 and was a first lieutenant engineer in charge of propulsion of a Corvette. Mr. Gillies was discharged in 1941 and accepted a position with the Foster Wheeler Corp., Dansville, N. Y. He held a temporary position with the Goodyear Tire & Rubber Co., Akron as a product engineer and in 1951 became associated with the Biggs Boiler Works Co., Akron. Later he was employed by Jack & Heintz, Inc., Bedford, Ohio as a standards engineer. Mr. Gillies was also a member of ASME, a fellow, American Institute of Chemists, and a member of the Institute of Chemical Engineers (London).

Proposed for Membership
— Metals Branch AIME —

Total AIME membership on Mar. 31, 1953 was 18,475; in addition 1098 Student Associates were enrolled.

ADMISSIONS COMMITTEE

O. B. J. Fraser, Chairman; Philip D. Wilson, Vice-Chairman; F. A. Ayer, A. C. Brinker, R. H. Dickson, Max Gensamer, Ivan A. Given, Fred W. Hanson, T. D. Jones, G. W. Lutjen, E. A. Prentis, Sidney Rolle, J. T. Sherman, F. T. Sisco, R. L. Ziegfeld.

The Institute desires to extend its privileges to every person to whom it can be of service, but does not desire as members persons who are unqualified. Institute members are urged to review this list as soon as possible and immediately to inform the Secretary's office if names of people are found who are known to be unqualified for AIME membership.

In the following list C/S means change of status; R, reinstatement; M, Member; J, Junior Member; A, Associate Member; S, Student Associate.

California
Berkeley—McMillan, Donald J. (M) (R, C/S—S-M)
San Francisco—Matsuda, Takashi (A) (C/S—S-A)
San Marino—Gallaher, Robert A. (M) (C/S—A-M)

Connecticut
Bridgeport—Engstrand, E. Raymond (M)
Milford—Beatty, George R. (J)
New Milford—Eastman, Walter F. (M) (R, C/S—A-M)
Norwalk—Biers, Howard W. R. (M)
Suffield—Berry, Clayton D. (M)
West Hartford—Frost, Charles E. (M)
Wethersfield—Korten, Robert D. (J)

Delaware
Wilmington—Warren, Donald (J) (C/S—S-J)

District of Columbia
Washington—Caldwell, Herbert S., Jr. (A)

Illinois
Chicago—Barry, Richard E. (M)
Chicago—Kurtz, Norbert R. (J) (C/S—S-J)
Chicago—Mitchell, Joseph R. (M) (C/S—A-M)
Chicago—Nollinger, Philip L. (M)
Chicago—Walsh, William J. (M)
Winnetka—Day, John N. (J)

Indiana
East Chicago—Goldenstein, Adolph W. (J) (C/S—S-J)
Gary—Smith, Phillip H. (J) (C/S—S-J)
Hammond—Moyland, Frank J. (M)
Highland—Harvey, Wesley C. (J) (C/S—S-J)
Highland—Karlson, Edward J. (M)

Massachusetts
Indian Orchard—Low, Sidney (M) (C/S—A-M)

Michigan
Dearborn—Altstetter, John D. (J) (C/S—S-J)

Montana
Butte—Hames, Frederick A. (M) (C/S—A-M)

New Jersey
Essex Falls—Iredell, Charles V. (M)
Murray Hill—Lesnewich, Alexander (J) (C/S—S-J)
Runnemede—Smith, Raymond L. (M) (R, C/S—S-M)

New Mexico
Hurley—Lockart, Charles L. (M) (C/S—A-M)

New York
Great Neck—Watt, Archibald M. (M) C/S—A-M)
Rochester—Bishop, Robert W. (M) (C/S—A-M)
Scarsdale—Webster, William H. C. (M)

Ohio
Akron—Kline, Robert H. (A)
Cleveland—Forsyth, Henry J. (M) (R, C/S—J-M)
Columbus—Stephens, Frank M., Jr. (M) (C/S—J-M)
Lakewood—Ragland, Ford J., Jr. (J) (C/S—S-J)

Oregon
Portland—Holekvam, Donald B. (M) (R, M)

Pennsylvania
Belle Vernon—Mueller, Victor T. W. (M)
Charleroi—Cox, Jacob W. (M)
Morrisville—Bartels, Keith D. (M)
Morrisville—Miller, Emerson R. (M)
Pittsburgh—Buck, Dana C. (M)
Pittsburgh—Dorrance, William E. (M)
Pittsburgh—McCleskey, George H. (E.T.) (J)
Pittsburgh—Squier, William R. (M) (C/S—A-M)
Pittsburgh—Vadeboncoeur, James O. (M) (C/S—A-M)
Sharpesville—Norton, Benton W. (M)
Turtle Creek—Mauk, Harry G., Jr. (A) (R, C/S—J-A)

Tennessee
Oak Ridge—Harms, William O. (J) (R, C/S—S-J)

Texas
El Paso—Bradish, Ames D. (A)
El Paso—Kelly, William R. (M) (C/S—A-M)
El Paso—Vargas, Jorge R. (J)

Utah
Garfield—Winkel, Matthew J. (M)
Provo—Tofft, Leonard (M) (C/S—J-M)
Salt Lake City—Hardman, Vern C., Jr. (J)

Washington
Vancouver—Bergman, John L. (M)

West Virginia
Huntington—Matasich, Charles L. (A)
Huntington—Milby, Elmer (M)

Wisconsin
Madison—Adams, Stanley B. (M)
Waukesha—Hofmeister, William F. (M) (R, C/S—S-M)

British Columbia
Vancouver—Magnolo, Giuseppe (J) (C/S—S-J)
Vancouver—Parr, James G. (J)

Chile
Santiago—Gajardo Morales, Sergio (J) (C/S—S-J)

Germany
Essen—Wasmuht, Roland W. (A)
Rheinhausen—Ossmann, Salah A. (J)

Mexico
Federal District—Von Ziegler, Emilio N. (J) (C/S—S-J)

Norway
Kirkenes—Braaten, Tormod (A)

South Australia
Tusmore—Jackson, Norton (M) (C/S—A-M)

Turkey
Gaziantep—Tuzun, Ali A. (J) (C/S—S-J)

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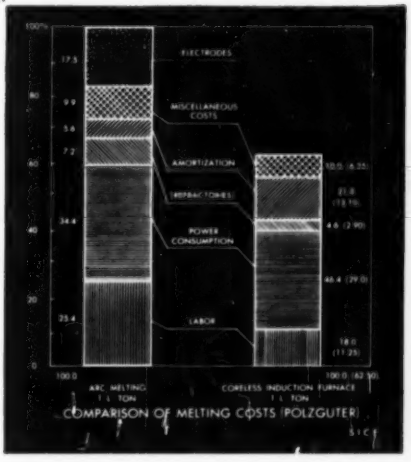

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